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Bulletin 1

DEPARTMENT OF THE INTERIOR

BUREAU OF MINES

JOSEPH A. HOLMES, DIRECTOR

THE VOLATILE MATTER OF COAL

BY

HORACE C. PORTER

AND

F. K. OVITZ



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THE VOLATILE MATTER OF COAL.

By HORACE C. PORTER and F. K. OVITZ.

INTRODUCTION.

SCOPE OF REPORT.

The various fuel investigations that were being carried on by the technologic branch of the United States Geological Survey were transferred by law on July 1, 1910, to a new federal bureau, the Bureau of Mines, that was authorized to continue the investigations and make public reports of the results. In consequence of this transfer, the following report is published as a bulletin of the new bureau.

The United States Geological Survey had been engaged in analyzing and testing coals, lignites, and other mineral fuel substances under authority given by act of Congress. This work, now centered at the experiment station at Pittsburg, Pa., had its beginning in the operations of the coal-testing plant erected at the Louisiana Purchase Exposition in St. Louis, Mo., in 1904. The results obtained at that plant showed that the work of determining the fuel value of the coals and lignites in the United States with a view to increasing efficiency in their utilization would be incomplete if it did not include systematic physical and chemical researches into the processes of combustion. Hence in its later investigations the Survey carried on such researches, concentrating attention on those lines of inquiry which promised results of greatest economic importance.

This bulletin is a report on an investigation of the volatile matter in several typical coals—its composition and amount at different temperatures of volatilization. As the investigation is still in progress and will doubtless include other coals than those already examined, the bulletin may be considered a preliminary report, stating the problems studied, the methods used, and the results thus far obtained.

SUMMARY OF RESULTS.

The investigation has already shown that the volatile content of different coals differs greatly in character. The volatile matter evolved from the younger coals of the West includes a large proportion of

carbon dioxide, carbon monoxide, and water, and a correspondingly small proportion of hydrocarbons and tarry vapors. The older bituminous coals of the Appalachian region yield volatile matter containing large amounts of tarry vapors and hydrocarbons, difficult to burn completely without a considerable excess of air and a high temperature. Coal of the western type, moreover, gives up its volatile matter more easily at moderate temperatures than coal of the eastern type. The volatile matter produced at medium temperatures is rich in higher hydrocarbons of the methane type, such as ethane and propane, which contain a larger proportion of carbon than is present in methane. These facts help to explain the difficulty of burning Pittsburg coal, for example, without smoke, the low efficiency usually obtained in burning high-volatile western coals, the advantage of a preheated auxiliary air supply introduced over a fuel bed, and the advantage of a furnace and boiler setting adapted to the type of fuel used. They bear directly also on the question of steaming "capacity" of coals for locomotives, the designing and operation of gas producers for high-volatile fuels, and the operation of coke ovens and gas retorts.

The results show further that certain bituminous coals of the interior and Rocky Mountain provinces give promise of good yields of by-products of coking, notably ammonia and high-candlepower gas, comparing favorably in these respects with the high-grade coking coals of the eastern province.

They show also that inert, noncombustible material is present in the volatile products of different kinds of coal to an extent ranging from 1 to 15 per cent of the coal.

DEFINITION OF VOLATILE MATTER.

The term "volatile matter" in coal, as distinguished from the term "volatile combustible matter," which is sometimes used, may be criticised on the ground that moisture is volatile and would therefore by implication be included in the term "volatile matter." However, the present investigation shows that the volatile matter exclusive of "moisture" contains considerable percentages of noncombustible materials, such as carbon dioxide, water, and nitrogen, and the application of the term "combustible" is therefore hardly justifiable. Less confusion seems likely to result from designating the volatile matter exclusive of moisture "volatile matter," even though moisture is also volatile, than would result from applying the term "combustible" to material which is one-tenth to one-third noncombustible.

OBJECT OF INVESTIGATION.

Methods for the quantitative estimation of volatile matter have not entered into the present investigation, the object of which has been rather to determine the composition of the volatile matter evolved at

different temperatures from different types of coal and to show the influence of this factor on efficiency in the use of coal.

BEARING ON BOILER-FURNACE OPERATION.

If a furnace produces 15 pounds of flue gases from 1 pound of fuel, and if these gases, by reason of improper firing or furnace setting, contain unconsumed combustible gases to the extent by volume of 0.5 per cent of CO, 0.5 per cent of H, and 0.4 per cent of CH₄, the theoretical loss through these materials is approximately 10 per cent of the heating value of the coal. For every 0.1 per cent of CO in the flue gas the efficiency is lowered theoretically 0.47 per cent; for 0.1 per cent of H, 0.48 per cent; and for 0.1 per cent of CH₄, 1.52 per cent.

In the report entitled "A study of four hundred steaming tests"^a it is shown that in 48 tests made at the St. Louis fuel-testing plant of the Geological Survey, as the CO in the flue gas increased from 0.3 to 0.7 per cent the efficiency dropped from 65 to 57 per cent—about 2.0 per cent for each 0.1 per cent of CO, or four times the above-stated theoretical drop. The amounts of H and CH₄ in the flue gas were not determined in these tests. In the discussion of the relation between efficiency and CO, however, the following statement is made: "We therefore reach the inevitable conclusion that at least two-thirds of the large drop in code 'boiler efficiency' with rise of CO is due to incomplete combustion losses not represented by CO, so that high CO is a decided danger signal." Very few records of determinations of hydrocarbons or hydrogen in flue gases are to be found, presumably because of the analytical difficulty in determining such small quantities. In the report of the Manchester (England) Smoke Committee, 1895,^b are given some flue-gas analyses showing in certain tests of hand-fired furnaces 0.25 to 0.40 per cent of CH₄ and 0 to 1.0 per cent of H. Recent tests at the Geological Survey's plant on small house-heating boilers using Pittsburg slack coal have given in the flue gases as high as 2.8 per cent of CO and nearly 1.0 per cent each of H and CH₄, the conditions in these tests being, however, extremely unfavorable to complete combustion and not parallel to those in larger plants. There seems to be a strong probability, nevertheless, that in the greater number of smoking chimneys volatile combustible gases are escaping which carry from 3 to 10 per cent of the total heat value of the fuel; and it is easy to see the importance of improving methods for the analytical determination of methane and hydrogen in flue gas and of taking samples which are less diluted with air by leakage through the furnace setting and in which the CO, CH₄, and H would therefore appear as larger percentages.

^a Bull. U. S. Geol. Survey No. 325, 1907, pp. 28, 65.

^b Cited in Bull. U. S. Geol. Survey No. 334, 1908, p. 14.

E. J. Constam and P. Schläpfer,^a in an extended series of boiler trials of European coals, have shown in a striking manner the effect of the amount and composition of the volatile matter on the boiler efficiency. They found in the flue gas from one coal 3.5 per cent of CO and 1.5 per cent of H, which caused an efficiency loss of 17.2 per cent.

Several factors may enter into the unaccounted-for losses in the heat balance of a steaming test. For example, it is likely that more heat is carried away by the flue gases (including steam) than is usually charged to them, for the reason that their specific heat probably increases with temperature; and this factor is of special importance for low-grade, high-moisture, high-oxygen coals. Another factor is the loss of solid combustible material in the chimney gas, or "sparking," which also is more prevalent with low-grade fuels. That the most important "unaccounted-for loss," however, is likely to be found in the unburned volatile gases which are allowed to escape through lack of sufficient combustion space or of a sufficient supply of heated air, the above considerations are intended to show. The present paper indicates that in order to obtain high efficiency the furnace must be adapted to the fuel and that the character, as well as the amount, of volatile products is of decided importance. The following results of steaming tests made at the Geological Survey's plant, compared with some results from the present investigation on the character of the volatile matter in the coals used, will further illustrate this point:

TABLE 1.—*Steaming tests of various coals.*

Coal No.	Volatile matter in dry coal (per cent).	Combustible gas at 600°. ^a	Heavy hydro-carbon gases at 600°. ^a	Steam test No.	Boiler efficiency (per cent).	Unaccounted-for loss (per cent).	CO in flue gas (per cent).	Rate of firing. ^b
W. Va. 11.....	20.8	60	5.6	56	68.3	5.4	0.05	18.1
Pa. 19.....	32.9			498 308	63.9 65.6	8.7 7.1	.04 .02	20.4 21.0
Lab. 3.....	30.4	75	8.5					
Ill. 19.....	33.0	108	10.5	160-63	63.7	12.5	.19	19.0
W. Va. 13.....	32.6	116	13.8	179-80	68.1	11.1	.14	15.7
Wyo. 4.....	45.4			399	56.4	15.9	.07	24.0
Lab. 18.....	43.4	142	19.4					
Wyo. 1.....	43.8	165	15.1	63	54.9	11.2	.25	22.7
Tex. 4.....	41.6			291 298	59.9 51.2	6.9 17.6	.0 .0	23.1 35.2

^a Volume in cubic centimeters from 10 grams in ten minutes, laboratory test.

^b Pounds of dry coal per square foot of grate surface per hour.

W. Va. 11. Pocahontas bed, "smokeless" coal.

Pa. 19. Pittsburg bed, Westmoreland County.

Lab. 3. Pittsburg bed, Connellsville, Fayette County, Pa., similar to Pennsylvania 19.

Ill. 19. No. 7 seam, Zelgier, Franklin County.

W. Va. 13. "No. 2 gas coal," Page, Fayette County.

Wyo. 4. Bituminous coal, Hanna, Carbon County.

Lab. 18. Bituminous coal, Diamondville, Uinta County, Wyo., somewhat similar to Wyo. 4.

Wyo. 1. Subbituminous, Sheridan district, 22 per cent moisture.

Tex. 4. Lignite, Wood County, 36 per cent moisture.

Inasmuch as the values given for unaccounted-for losses in a steaming test embody all the errors of the other determinations, little dependence can be placed on any relation shown under that heading. The relations shown, however, under boiler efficiency, CO, and rate of firing, as compared to ease of liberation of volatile gases, are instructive. Wyo. 1, for example, compared with W. Va. 11, was burned with low efficiency, high CO, and high unaccounted-for loss, while in the laboratory test its gases were liberated much more easily. The rate of firing is an important factor. Coals of low heating value must be fired at a high rate in order to maintain a given boiler capacity, and as these coals are commonly high in volatile matter, the danger of loss of volatile gases is correspondingly great. W. Va. 13, in the table above, was fired at a low rate, which may account for the fact that it shows high efficiency, even though its volatile gases are easily liberated.

BEARING ON SMOKE PRODUCTION.

Visible smoke consists of solid carbon particles and solid or liquid hydrocarbon particles or "tar vapors." Both result from incomplete combustion of the volatile products of the fuel. The carbon of the smoke is not derived from free carbon in the fuel, but is deposited by the cooling of hot dissociated hydrocarbon gases. Flame is a phenomenon accompanying the chemical union of certain gases, one of which is usually oxygen;^a and the incandescent solid particles make a flame visible. If some of these particles in the flame are carbon, formed by the dissociation of hydrocarbons, luminosity results; and if the temperature of these particles is reduced below the point at which they combine with oxygen, or if sufficient oxygen is not at hand to effect the union, they fail to unite with oxygen and pass off as solid carbon in smoke. These principles are stated very clearly by L. P. Breckenridge in "How to burn Illinois coals without smoke."^b When a coal produces rich volatile gases, bearing large amounts of heavy hydrocarbons, a combustion space must be provided sufficiently large to allow the flame to be burned out before striking cool surfaces; and the flame must have an adequate supply of air at an adequate temperature if it is to be burned out in time. Furthermore, when rich volatile products distill rapidly from a coal at medium and low temperatures, they must be taken care of by increased combustion space or by decreased rate of firing. The essential requirements of smokeless combustion are therefore three—(1) sufficient combustion space, (2) sufficient air at a high temperature, and (3) sufficient thorough mingling of gases and air—these three conditions to be adapted to the type of fuel and the nature of its volatile products. Tabulated data and comparisons are given elsewhere in this paper (pp. 32-41) to show the bearing of the results on the smoke-producing tendencies of the coals tested.

^a Lewes, V. B., The luminosity of coal-gas flames: Proc. Roy. Soc., vol. 57, 1895, p. 450.

^b Bull. 15, Univ. Illinois Eng. Exper. Sta., p. 7.

BEARING ON LOCOMOTIVE FIRING.

For locomotive use that coal is most desirable which will give the highest boiler efficiency when burned so as to produce the required high evaporative rate (11 to 14 pounds of water per square foot of boiler heating surface per hour). Reduction of boiler efficiency under these conditions may be due to several causes, chief among them being loss of sensible heat, loss of cinders, and loss of combustible gases through the stack. That the last-named factor is of large influence it seems safe to conclude from the general rule that high rate of firing involves increase of unconsumed combustible gases in the flue gas. In view of the high rate of firing and the small combustion space in a locomotive, the bearing of the nature of the volatile products of the fuel on the completeness of their combustion under these unfavorable conditions may readily be seen.

BEARING ON GAS-PRODUCER, COKE-OVEN, AND GAS-RETORT OPERATION.

At the top of the fuel bed in a gas producer the volatile products of the fuel are distilled. In many types of producer these products are distilled at medium and low temperatures, and those of some high-volatile fuels contain such large amounts of tar and heavy hydrocarbons that it is necessary to draw them downward through the hot fuel and convert them into permanent gases in order to avoid losses through the deposition of tar and soot. The value of a knowledge of the nature of the volatile products from different fuels is evident.

The carbonization of coal in coke ovens and gas retorts is chemically an exceedingly complex process. It involves the distillation of volatile products at all temperatures from 100° to 1,200° C. The composition of the final volatilized product is determined not only by the temperature within the distilling substance, but also more largely by the temperature of the region through which the products pass. This composition is a resultant of the distillation of many coal particles, each probably at a different temperature, and of the time and temperature conditions to which the first products are subjected after leaving their original state of combination in the coal substance. The bulk of the mass heated and the rate of supply of heat from without determine the time and temperature conditions to which the products are subjected. In the study of industrial coal-carbonization processes, coke ovens, horizontal gas retorts, and vertical retorts, therefore, a knowledge of the variation among coals in the nature of their resultant volatile products at different maximum temperatures should aid in fixing the most favorable working conditions. The results presented in this paper show that the first products when coal is gently heated without access of air are CO,

and saturated paraffin hydrocarbons. From coking and gas coals the latter are formed in abundance, and it is from their decomposition at high temperatures that the various constituents of coal gas and coal tar are formed. Highly oxygenated coals produce large quantities of CO_2 and CO .

NATURE OF COAL SUBSTANCE.

The question whether coal is a mixture of free carbon, hydrocarbons, and complex oxygenated molecules or a uniform molecular substance of definite composition has been the subject of many investigations but still remains unsolved. The most promising method of attacking the problem seems to be that of extraction with solvents or modification of the substance by treatment with chemical reagents, together with systematic microscopical examination. Such investigations are now being carried on in the laboratories of the Bureau of Mines. It is hoped that with the results of these studies the data on destructive distillation afforded by the present investigation may be coordinated in such a way as to aid in throwing light on the problem.

EFFECT OF OXYGEN IN COAL ON CALCULATION OF HEAT VALUE FROM ULTIMATE ANALYSIS.

Oxygen, as well as ash, moisture, and nitrogen, exercises an anti-calorific influence on coal. Not only is it a diluent, or so much "dead weight," like the other three constituents mentioned, but it also neutralizes or renders ineffective an equivalent combining weight of carbon, hydrogen, or sulphur. In some coals this neutralizing action may be accompanied by the development of heat, for it is known that in destructive distillation exothermic reactions take place.^a But oxygen is nevertheless in combination in the coal substance and must therefore be considered as rendering inert a certain portion of the fuel elements. In all the applications of fuel the first stage in the decomposition is probably the distillation of the volatile products, and as hydrogen has a higher calorific value than an equivalent weight of carbon, the distribution of oxygen between carbon and hydrogen in the volatile products has a direct influence on the calorific value of the fuel. Available hydrogen has greater value than available carbon. P. L. Dulong's method of calculating heat value reduces the available hydrogen by an amount equivalent to all the oxygen, and therefore gives too low a value in coals which distribute a large proportion of their oxygen to carbon rather than to hydrogen. The experimental results given in this paper show that in certain low-grade, highly oxygenated coals nearly two-thirds of the oxygen

^a Constam, E. J., and Schläpfer, P., Jour. Gasbel., 1906, pp. 741, 774.

appears in the volatile products in union with carbon, and that this accounts largely for the difference between the determined heat value and that calculated by Dulong's method. (See pp. 43, 44.)

RELATED INVESTIGATIONS BY OTHERS.

E. Börnstein^a has carried out elaborate experiments on the distillation of German bituminous and brown coals at temperatures up to 450° C. He obtained tars with no solid aromatic hydrocarbons and gases containing large amounts of CO₂ and homologues of methane.

E. J. Constam and E. A. Kolbe^b distilled at high temperatures a number of bituminous coals of varying composition and obtained from those high in oxygen large amounts of CO₂ and CO and correspondingly less of hydrocarbons.

L. Vignon^c distilled at 900° C. five coals of varying known oxygen content and attempted to establish a ratio between the amount of oxides of carbon and the oxygen in the coal.

S. W. Parr and C. K. Francis^d have distilled Illinois coal (4 to 5 pounds) at 200° to 425° C. for two to three hours in an inert atmosphere and obtained from 3 to 4.5 per cent of water of composition, a small amount of oil, and 600 to 700 cubic feet per ton of gas high in CO₂, illuminants, and "methane." Homologues of methane are not mentioned in the report of this work, but are probably included under "methane."

A. H. White, with his assistants, F. E. Park and W. A. Dunkley,^e has studied the low-temperature distillation of three typical American coals with regard to its bearing on the manufacture of illuminating gas and its effect on the quality of the coal. He distilled 50 to 75 grams for six to eight hours at 300° to 500° C. and obtained small quantities of liquid and oily distillates with 1,200 to 2,800 cubic feet per ton of gas rich in methane and ethane. The following tabulated statement of results is given:

TABLE 2.—*Gas obtained by distillation of three typical American coals.*

	Pitts- burg, Pa.	Bay City, Mich.	Zeigler, Ill.
Volume of gas (cubic feet per pound of coal).....	1.42	1.15	0.63
Average composition of gas:			
CO ₂	2.9	16.2	13.1
Illuminants.....	2.2	4.1	1.6
CO.....	6.2	5.0	5.8
H.....	26.3	16.4	13.9
CH ₄	47.0	37.8	38.0
C ₂ H ₆	13.2	11.8	19.5
N.....	2.7	9.1	7.8

^a *Zeitschr. angew. Chemie*, vol. 17, 1904, p. 1520; *Jour. Gasbel.*, vol. 49, 1906, pp. 627, 648, 667; *Jour. Soc. Chem. Ind.*, vol. 25, 1906, pp. 213, 583.

^b *Jour. Gasbel.*, vol. 51, 1908, p. 669.

^c *Bull. Soc. chim.*, 4th ser., vol. 3, 1908, p. 109.

^d *Bull. 24, Univ. Illinois Eng. Exper. Sta.*, 1906; *Trans. Am. Inst. Min. Eng.*, 1908, p. 1158.

^e *Am. Gas Light Jour.*, vol. 89, 1908, p. 621.

The heating power of the solid residue was greater than that of the coal, but the coking property was destroyed. The manner and the location of temperature measurements are not stated.

R. T. Chamberlin, of the United States Geological Survey, has made a study^a of the gases liberated from coal at ordinary temperature by crushing in a vacuum, and of the gas evolved from coal dust at somewhat higher temperatures (200° to 450° C.). At 450° he found a continuous evolution of gas through one and one-half hours and longer, which he ascribes largely to chemical changes in the coal substance induced by heat rather than to liberation of imprisoned gas. The gas at 450° consisted of 60 to 70 per cent of methane, 12 to 18 per cent of ethane, 5 to 6 per cent of heavy hydrocarbon vapors (benzene, ethylene, etc.), and small percentages of carbon dioxide, carbon monoxide, and sulphurous gases. Chamberlin's results are applied to an investigation of the causes of mine explosions, the relative explosibility of weathered and fresh dust, and the method of accumulation of gas in mines. At temperatures below 350° he obtained only very slight percentages of ethane in the gas, and concludes that in the imprisoned gases of the coals studied (Pennsylvania and West Virginia) higher hydrocarbons of the methane series are not present to any appreciable extent.

EXPERIMENTAL PLAN OF PRESENT INVESTIGATION.

It has been the plan in the present investigation to extend such studies as those described above to several types of American coals and to a greater variety of experimental conditions of such kind as to indicate the behavior of these coal types in their industrial applications. It has been borne in mind throughout that industrial conditions can scarcely be duplicated in the laboratory, and that any selected set of laboratory conditions (of temperature and of quantity, for example) must be considered as corresponding to merely a portion of the varied conditions which exist simultaneously in an industrial application. The laboratory method, however, permits more exact measurements and presents the further advantage that a few definite and controllable conditions can be selected from among the many indefinite ones of industrial practice, and the effect of these conditions can be determined with more certainty. It is not claimed that the experimental data of this investigation show absolute industrial yields either in quantity or composition of products, but it is held that they afford a comparison of various coals under fixed conditions, and also that they justify conclusions as to the probable industrial behavior of the different coals, in furnace, coke oven, or producer.

^a Notes on explosive mine gases and dusts: Bull. U. S. Geol. Survey No. 383, 1909.

For comparing coals as to the composition and the rate of formation of their volatile products at a number of fixed temperatures, 10-gram samples of air-dried powdered coal were heated in an inert atmosphere in a platinum retort. For comparing coals as to their yields of by-products of coking, under one fixed set of conditions approximating those of industrial practice, tests were made on 400-gram samples of crushed coal as received, heated in a cast-iron retort. Direct weighing of tar and of water of composition and the determination of ammonia were capable of greater accuracy in the 400-gram tests than in the 10-gram tests. Nine different coal types were used, ranging from a low-grade subbituminous coal from Wyoming to the high-grade Connellsville and Pocahontas coals of the Appalachian region.

METHODS.

COLLECTION AND PREPARATION OF SAMPLES.

The samples of coal used in these investigations were with one exception mine samples collected by representatives of the United States Geological Survey in the manner prescribed for the work of the fuel-testing plant.^a The one exception was coal No. 23 from a mine near Harrisburg, Saline County, Ill., this sample being a car sample of screened coal (2-inch to 3-inch size) taken by a representative of the Illinois State Geological Survey from a car at the mine, one day after it had been mined.

Samples of about 100 pounds each, representing the entire seam as mined, were shipped to the laboratory in canvas bags or (in case of coals particularly subject to deterioration from exposure) in tightly closed kegs or barrels.

At the laboratory the entire sample was crushed to $\frac{1}{2}$ -inch size and a portion of 8 to 10 pounds was taken by quartering for air drying. From this air-dried portion a 500-gram sample was taken by quartering, pulverized to 60-mesh size in the closed ball mill, and bottled in a "lightning" fruit jar for analysis. The bulk of the original sample was placed in a galvanized ash can having a tightly fitting cover, and the balance of the air-dried sample not pulverized was placed in a tin milk can with a tight cover sealed with tape. The time that elapsed between mining the coal and placing it in covered containers varied from three to five weeks; during this time there was undoubtedly some loss in mine moisture and possibly slight changes in the coal substance due to oxidation. The subbituminous coals, however, which are high in moisture and more subject to change, were kept in tightly closed kegs and probably changed very little.

^a Bull. U. S. Geol. Survey No. 280, 1906, p. 17.

The proximate analyses and sulphur, nitrogen, and calorimetric determinations were made on the air-dried powdered samples according to the methods of the fuel-testing plant of the Survey.^a

TESTS IN IRON RETORT.

APPARATUS.

For the purpose of comparing coals in respect to their yields of by-products of coking as well as in the composition of their volatile products, a cast-iron retort holding somewhat less than a pound of coal was used, together with apparatus as illustrated in Plate I, A.

The bottles *a*, *b*, *c*, and *d* contain dilute sulphuric acid; the tower *e* and condenser *f* contain glass beads over which water flows from the funnel *g*; the upper portions of *f* contain absorbent cotton to collect the last traces of tar. It was found by placing two drops of normal acid in *d*, with a little cochineal as indicator, that no ammonia passed through the acid in *c*. The bottle *h* contains 30 per cent potassium hydroxide solution and the tower *i* holds glass beads wet with the same solution from funnel *j*. The gages *k* indicate the gas pressure at the beginning and end of the train. Temperatures in the furnace and inside the iron retort were read by means of thermocouples connected through a cold junction bath (*l*) and switch to the millivoltmeter (*m*).

Figure 1 shows a vertical section through the furnace viewed from the side. This furnace was designed and constructed on the plan of those used by A. L. Day in the geophysical laboratory of the Carnegie Institution, Washington, D. C.^b The outer cylindrical jacket (A) is of ordinary fire clay and the inner one (C) of "magnesite," a highly refractory material composed chiefly of magnesium oxide. The inner cylinder rests upon small fire-clay blocks, and the space (B) between the two cylinders is filled with light calcined magnesia. For convenience in winding the heating coils, the inner cylinder is in two sections placed end to end. The heating coils (D) are of pure nickel wire, No. 13 Brown & Sharpe gage, and are cemented to the inside of the inner cylinders by means of "magnesite" powder wet with water and a little gum tragacanth. The strands of wire are about 2 millimeters apart. For winding the coils a cylindrical arbor of hard wood is used, which is slightly smaller than the inside diameter of the furnace and is made up of wedge-shaped sections, so as to be collapsible and readily removed. The arbor, after covering with paper, is wound with the wire, coated with cement, and placed in the furnace cylinder, and the annular space is packed with cement. After drying thoroughly, the arbor is removed and a final surface layer of cement is coated over the coils.

^a Prof. Paper U. S. Geol. Survey No. 48, 1906, pp. 178-190.

^b Day, A. L., and Allen, E. T., *Am. Jour. Sci.*, 4th ser., vol. 19, 1905, p. 93.

A furnace made thus with nickel wire stands from 50 to 75 heats at $1,100^{\circ}\text{C}$. without renewal of the coils. The cement covering of the coils must, however, be kept intact by frequent patching to insure long life of the wire. A direct current of 220 volts was used, the strength varying from 18 to 25 amperes according to the variation of the resistance with the temperature.

Figure 1 shows also a vertical section of the iron retort viewed from the side. When the inside thermocouple is used, it is found convenient to add the coke through the delivery pipe after screwing it in

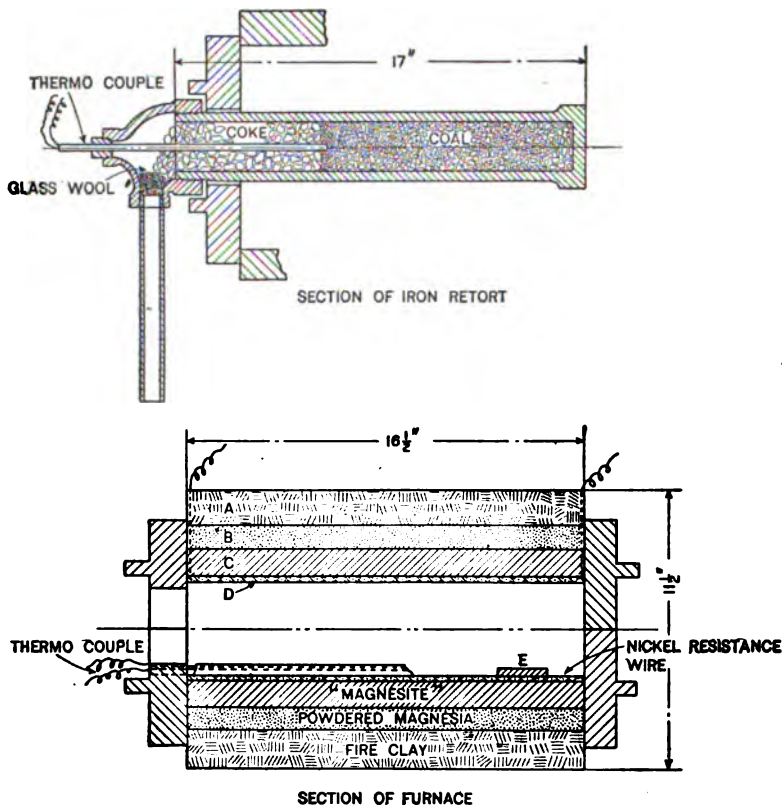


FIGURE 1.—Section of iron retort and section of large electric furnace.

place. The retort burns away on the surface and can be used for only 6 to 8 heats at $1,100^{\circ}$.

It was found impracticable to have the threaded joint of the retort inside the furnace and maintain it gas tight or to constrict the neck of the retort near the joint because of difficulty in removing the coke. If the threads are well made and kept lubricated with graphite there is no difficulty in keeping the joint gas tight. The walls of the retort were made heavy (one-half inch) so as to withstand better the scaling effect of the high heat. The conditions as to rate of heating

were therefore somewhat different from those of a coke oven or gas retort where the coal is charged directly into the heated chamber. The difference, however, is due more to loss by radiation through the large exposed surface of the retort outside of the furnace than to the presence of the iron wall between the coal and the furnace. The iron wall of the retort transmits heat to the coal rapidly, being a better conductor of heat than the layer of partly coked coal which forms next to the walls of a coke oven when the charge is introduced. The temperature less than 4 inches from the wall of a retort coke oven has been found to remain below 200° C. for five hours after charging.

The radiation, however, from the front and neck of the iron retort lowers the temperature of those portions to such a degree that tar vapors and heavy hydrocarbon gases pass through with less breaking down than in coke ovens or gas retorts; consequently the yields of tar and heavy paraffin hydrocarbons are greater in the laboratory tests than in commercial practice. The front part of the retort is filled with coke, in pieces of about $\frac{1}{4}$ -inch size, for the purpose of preventing the crowding forward of coal and tar. This coke does not become hotter than 600° C., on account of the cooling effect of radiation on the front of the furnace and retort. It is important to have the coal occupy the same space in the retort and be in the same relative position in the furnace in all tests, for the gas yield is materially influenced by these conditions.

The thermocouples and millivoltmeter used were standardized at the beginning of the investigations by the Bureau of Standards, Washington, D. C., and found to be correct within 5° C. at temperatures between 400° and 1,600° C., the cold junctions being at 25° C. They were standardized also after the completion of these investigations, at the physical laboratory of the United States Geological Survey, by means of the melting points of pure zinc and pure copper, and were again found correct within the limit mentioned.

The gas meter used (*p*, Pl. I, *A*) was Goodwin's experimental gas meter of 0.1 cubic foot capacity per revolution, furnished by the American Meter Company. The correct water level in the meter was established by aspirating about 0.5 cubic foot of air through the meter and determining its volume by the weight of water displaced in the aspirator bottles. For collecting the gas a 55-gallon barrel (*r*, Pl. I, *A*) which had been rendered gas tight by a coat of shellac was used. The gas was collected by displacement of water previously saturated with gas.

OPERATION.

The furnace having been brought to a temperature of 1,070° to 1,080° C. at its center, the retort charged with 400 grams of coal and 100 grams of coke, as shown in figure 1, and the train of by-product

apparatus made ready, the retort was quickly placed in the furnace and the fire-clay doors were replaced. Connection was then made with bottle *a* (see Pl. I, *A*) and temperature readings were taken at intervals of five minutes. The temperature of the furnace is lowered to 720° to 750° by the introduction of the retort, but after about five minutes begins to rise again at the rate of 5° to 6° per minute. Gas begins to be evolved about five minutes after the retort is placed in the furnace. Bottles *a*, *b*, *c*, and *d*, and the tower *e* were weighed with their connections and contents before and after the test. The gain in weight, together with a correction for water evaporated (based on the volume of gas and its temperature), was taken as tar, water, and ammonia. These weighings were made with an accuracy of 0.2 gram (0.05 per cent on the coal sample used). An approximate separation of tar and water was made, but the method was necessarily crude and no claim is made for accuracy greater than 1.0 per cent. After all the tar and liquor from bottles *a*, *b*, *c*, and *d* were collected the bulk of the aqueous liquor was decanted and the tar washed thoroughly with a measured quantity of water in six or eight successive portions. The water from tower *e* was used as part of the wash water. These washings were added to the decanted liquor and the whole was measured. The tar was then distilled to 180° C. and the water in the distillate was measured. The total measured amount of water less the wash water added was subtracted from the total tar and water weighed, and the difference was taken as dry tar. For ammonia determination the aqueous liquor and washings were made up to definite volume and aliquot portions were distilled with alkali, the ammonia was received in excess of normal H_2SO_4 and the surplus acid was titrated with normal NaOH and cochineal. The degree of washing necessary to remove all ammonia from the tar was determined by analyzing successive washings, and an excess over this amount was used.

The water in funnel *g* (100 cubic centimeters) and the KOH solution in funnel *j* were admitted to the towers below in small successive portions during the test. The KOH liquor was collected after the test, made up to definite volume and its content of CO_2 determined by evolution with acid and absorption in weighed potash bulbs according to Hillebrand's method.^a A blank was run on the original KOH solution and correction made accordingly.

H_2S was determined in the KOH liquor by acidifying a small portion diluted to 400 or 500 cubic centimeters and titrating with decinormal iodine solution.

The gas was measured by the meter, the collecting reservoir being used merely for the purpose of obtaining a true average sample.

^a Hillebrand, W. F., The analysis of silicate and carbonate rocks: Bull. U. S. Geol. Survey No. 305, 1906, p. 150.

The air in the apparatus (about 0.2 cubic foot) at the beginning of a test was allowed to mix with the gas and the average analysis, corrected to an air-free basis, all the oxygen present being assumed as due to air. The meter reading, however, represents the true yield of gas, for the same volume remains in the apparatus after the test as before. There is a small error in obtaining the average analysis, as the gas remaining in the apparatus at the close is not of the average composition. Successive samples of gas were taken during the progress of the test, through a T connection between the meter and reservoir.

The coke was removed from the retort and weighed to 0.5 gram. A small amount of tar (less than 0.5 per cent) remains on the walls of the delivery pipe.

By means of the gages (*k*, Pl. I, *A*) a suction of about 2 inches of water was maintained beyond the meter and the gas pressure in bottle *b* was observed. This pressure varied from 8 to 25 inches of water and served incidentally to indicate leaks in the apparatus. The test was continued until gas practically ceased to be evolved—that is, until less than 0.01 cubic foot per minute was produced. The time required to reach this point varied somewhat with the coal but was usually about one hour. The inside temperature (at the surface of the coal) usually attained 800° C. The furnace temperature was not allowed to rise over 1,100° C.

TESTS IN PLATINUM RETORT.

APPARATUS.

For the purpose of comparing coals in respect to the quantity and composition of their volatile products at different temperatures, a small platinum retort having a capacity of about 150 cubic centimeters was used, the retort being provided with platinum cover, delivery tube, and inlet tube. The arrangement of the apparatus is illustrated in Plate I, *B*, and the detail of furnace and retort in figure 2.

The retort (*a*, Pl. I, *B*) was connected by means of heavy rubber tubing protected by asbestos to a 6-inch U tube and 6-inch straight tube, both tubes being filled with absorbent cotton and inclosed in air baths (*b*) maintained at 110° C. Following the tubes containing cotton were two 5-inch U tubes containing CaCl_2 (*c* and *d*), a protecting tube containing the same (*e*), and finally a pair of 5-liter aspirator bottles (*f*) for collecting gas. The water in the bottles had been previously saturated with gas. One bottle was graduated for measurement of the gas and contained a thermometer. The cotton tubes were previously dried in a current of air at 110° C. Connected to the inlet tube of the retort was a 20-inch straight tube containing

CaCl_2 (*h*) and a 12-inch tube containing soda lime (*g*) for drying and purifying the air or nitrogen entering the retort. Nitrogen for these tests was prepared by drawing a slow current of air first through

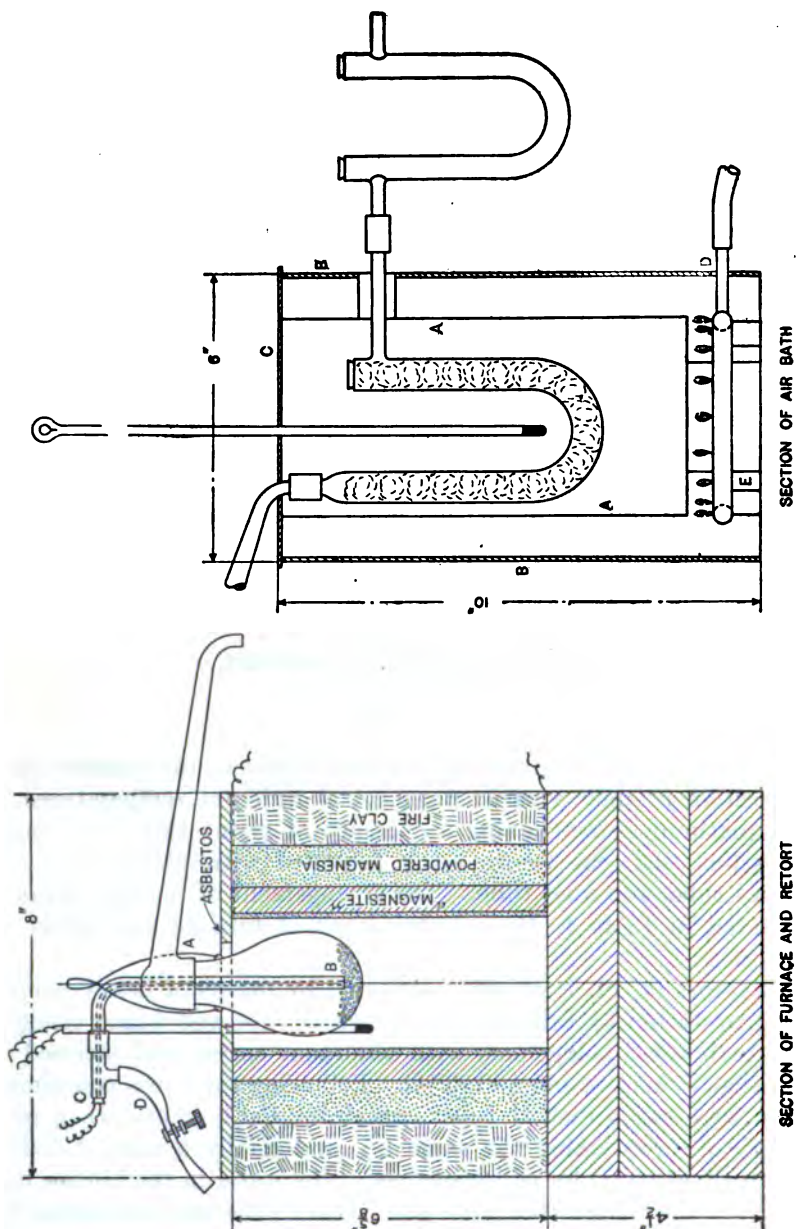


FIGURE 2.—Section of small electric furnace and platinum retort, and section of air bath.

alkaline pyrogallol solution in two wash bottles and then over heated copper gauze in a combustion tube. The nitrogen prepared in this way contained from 0.2 to 0.4 per cent of oxygen.

As shown in figure 2, the small vertical electric furnace was similar in construction to the large furnace used in the 1-pound tests, the dimensions differing, however, as indicated. The retort has a ground joint at A, which can be kept gas tight if shaped by a hard-wood form and polished with fine emery cloth before each test. The inner end of the platinum inlet tube is 10 millimeters above the bottom of the retort and the junction of the inside thermocouple (B) 5 millimeters above the bottom. It is important that the position of this thermocouple be kept the same in parallel tests, for a slight difference in its elevation materially affects the temperature reading. The wires of the couple were insulated by fine porcelain tubes, but the junction itself was bare. At C the wires of the thermocouple passed through a cork stopper and a coating of sealing wax; D is the inlet for dry nitrogen or air. The retort was placed in the furnace always at the same height and the temperature of the furnace was taken at the level of the bottom of the retort next the furnace wall.

In figure 2 is shown also the air bath for heating the tubes containing cotton. The inner cylindrical chamber (A) was of tin and was surrounded by a jacket of sheet asbestos (B). The cover (C) was of asbestos. The ring burner (D) encircles the block (E) which supports the bath.

OPERATION.

It was found advisable, on account of the necessity of aspirating in the tar and water determinations, to run the tests for gas separately. For the gas tests, therefore, the platinum retort was connected directly to the gas-collecting bottle, the tar and water being disregarded. The furnace temperature having attained constancy at the desired point, 10 grams of air-dried powdered coal was placed in the retort, the joint was made tight, and the air was displaced by nitrogen, a volume of nitrogen being aspirated equal to three times the capacity of the retort. This amount of aspiration was found to reduce the oxygen to less than 1.0 per cent in the atmosphere of the retort. Connection was then made to the gas-collecting bottle, the retort lowered into the furnace, and the time noted. Temperature readings were taken at one-minute intervals inside and outside of the retort and when the desired temperature in the coal (200° below the furnace temperature) was reached the retort was quickly removed and cooled by immersing in water. A thorough mixture of the gas and nitrogen was obtained by passing them back and forth three times from the gas-collecting bottle through the retort to the nitrogen reservoir. The gas was measured, its temperature taken, and an average sample analyzed. Although a certain proportion of the nitrogen present is undoubtedly produced by the coal, all analyses were calculated to a nitrogen-free basis to eliminate the effect of dilution with the added nitrogen.

In the tar and water determinations the gas was disregarded. After the retort was filled with nitrogen, the drying train was connected and the retort lowered into the furnace, the pinchcock at D (fig. 2) being closed. When the inside temperature had reached the desired point the retort was removed and cooled as before and 4 liters of dry air was drawn through it. During the aspirating process the platinum delivery tube was warmed with a small flame to drive out all water. The gain in weight of the retort cap and delivery tube was determined and, together with the gain in the cotton tubes, plus a correction for light oils, was taken as tar. The gain in the calcium chloride tubes less the correction for light oils was taken as water. In tests at 900° C. and higher it is necessary to use two cotton tubes in order to retain all the tar, though only one is shown in figure 2. As oils volatile below 110° C. are carried over into the calcium chloride tubes and weighed as water, a correction of 5 per cent of the tar weighed is applied, this being based on the maximum percentage of light oil in coke-oven tar according to Lunge.^a The tar and water separation is therefore approximate only, and no claim is made for greater accuracy in the separation than 1 per cent on the coal.

METHODS OF GAS ANALYSIS.

CARBON DIOXIDE, OXYGEN, ILLUMINANTS, AND CARBON MONOXIDE.

For the determination of carbon dioxide, oxygen, illuminants, and carbon monoxide the Hempel apparatus and the customary reagents were used; for CO₂, a 30 per cent solution of commercial NaOH, for oxygen alkaline pyrogallol solution containing 100 grams pyrogallol and 150 grams NaOH per liter, for illuminants fuming sulphuric acid (20 per cent SO₃), and for CO ammoniacal cuprous chloride solution according to Winkler's formula^b (250 grams NH₄Cl and 200 grams Cu₂Cl₂ in 1 liter, mixed just before use with one-third volume aqua ammonia of specific gravity 0.91). A record of the use of each pipette was kept by noting upon it the volume of gas absorbed during each determination. For the determination of more than 12 per cent of CO three pipettes were used successively. No greater accuracy than 0.2 per cent was attempted in these determinations.

BENZENE.

For the determination of benzene a large amount of experimental work was done in testing two methods, neither of which proved entirely satisfactory. The method of D. A. Morton,^c using sulphuric acid of specific gravity 1.84, was found to give approximately com-

^a Lunge, G., *Coal tar and ammonia*, 3d ed., p. 81.

^b Winkler, C., *Technische Gas Analyse*, 2d German ed., p. 77.

^c *Jour. Am. Chem. Soc.*, vol. 28, 1906, p. 1728.

plete absorption of the benzene from benzene and air mixtures in three minutes' shaking; but for coal gas containing various other hydrocarbons as well as benzene the method did not yield concordant results on the same sample or on samples prepared by closely similar methods and therefore was discarded. The method of L. M. Dennis and E. S. McCarthy,^a using an ammoniacal solution of ammonium nickel cyanide, was not found satisfactory. The absorption proved incomplete from known mixtures of benzene and coal gas in two minutes' treatment carried out as directed by the authors, and concordant results on the same sample could not be obtained.

In giving results of the gas analyses of the present investigation, therefore, no attempt has been made to differentiate the constituents absorbed by fuming sulphuric acid, designated as illuminants.

HYDROGEN.

In most of the tests hydrogen was determined separately by means of palladium asbestos, as recommended by F. C. Phillips,^b the method being that of Winkler.^c Palladium black was precipitated upon acid-washed ignited asbestos by the action of alkaline sodium formate on palladium chloride solution, the product being then washed, dried, and ignited at a moderate red heat. A mixture of the gas residue and air, equivalent to an oxygen-hydrogen ratio of at least 2:1 by volume, was passed slowly over the palladium in a small U tube maintained at 50° to 70° C. by a water bath. After this mixture was passed three times through the tube the contraction was complete and the gas was cooled by passing once through the tube immersed in water at room temperature. A water-jacketed burette was used for this determination and a Hempel pipette containing distilled water for receiving the gas after passage through the palladium tube. The palladium asbestos must be dried thoroughly after each combustion of hydrogen and must be carefully protected from contamination with acid fumes. According to Phillips hydrogen is completely burned under these conditions and hydrocarbons are not affected. The writers have verified this statement by tests with pure electrolytic hydrogen, and also on samples of natural gas, testing for the formation of CO₂.

METHANE AND ITS HOMOLOGUES.

For the determination of hydrocarbons of the general formula C_nH_{2n+2}, Winkler's method of slow combustion by a heated platinum spiral,^d as modified by L. M. Dennis and C. G. Hopkins,^e was used.

^a Jour. Am. Chem. Soc., vol. 30, 1908, p. 233.

^b Am. Chem. Jour., vol. 16, 1893, pp. 164-168.

^c Winkler, C., *Technische Gas Analyse*, 2d German ed., pp. 145-150.

^d *Idem*, pp. 155-157.

^e Hempel, W., *Gas analysis* (transl. of 3d German ed.), p. 138.

The combustion pipette was of glass. The spiral was made of about 10 inches of No. 24 platinum wire and the ends of the spiral were welded to somewhat larger platinum wires which, insulated by glass tubes, passed downward through the rubber stopper and were connected to the source of the electric current. It was found necessary to avoid the use of any baser metals than platinum in the fittings of the pipette, as a considerable error is introduced thus by oxidation of the metal.

The total contraction and CO_2 produced by the combustion having been determined, and the contraction due to hydrogen separately determined by palladium and subtracted, the result indicated the contraction and the CO_2 due to the burning of methane and its homologues. As shown by E. H. Earnshaw,^a the total volume of hydrocarbons of the type $\text{C}_n\text{H}_{2n+2}$ could then be calculated and also the value of n , showing the average composition of the hydrocarbon mixture. For a given amount of contraction and CO_2 , the total volume of $\text{C}_n\text{H}_{2n+2}$ is the same, regardless of the relative percentages of CH_4 , C_2H_6 , C_3H_8 , etc. Thus:

$$\text{C}_n\text{H}_{2n+2} + \left(\frac{3n+1}{2}\right)\text{O}_2 = (n) \text{CO}_2 + (n+1) \text{H}_2\text{O}.$$

If V = volume of $\text{C}_n\text{H}_{2n+2}$

$$(1) \text{ Contraction} = \left(V + \frac{3n+1}{2} V\right) - nV = V\left(\frac{n+3}{2}\right)$$

$$(2) \quad \text{CO}_2 = Vn.$$

$$\text{By combining (1) and (2) } V = \frac{2 \text{ contraction} - \text{CO}_2}{3}$$

and

$$n = \frac{\text{CO}_2}{V}$$

Without further data it is impossible to determine the relative percentage of any one of the hydrocarbons when more than two are present.

If we assume, however, that only methane (CH_4) and ethane (C_2H_6) are present, and find, for example, 1.25 for the value of n and 24.0 per cent for the total $\text{C}_n\text{H}_{2n+2}$, then ethane is 25 per cent of the total, or 6.0 per cent; if n is found to be 1.33 and the total 24.0 per cent, then ethane is 8.0 per cent.

^a Jour. Franklin Inst., September, 1898.

SUMMARY AND INTERPRETATION OF RESULTS.

COALS TESTED.

The following tables show the source and composition of the coals used for the tests herein described:

TABLE 3.—Source of coal samples used in tests.

Laboratory No. ^a	State.	Post-office.	Seam or bed.	Mine.	Date collected.
1	Illinois.....	Zeigler, Franklin County..	No. 7.....	Zeigler.....	June 23, 1907
3	Pennsylvania.....	Connellsville, Fayette County.	Pittsburg.....	Leisenring No. 1.	June 20, 1907
10	West Virginia.....	Page, Fayette County.....	No. 2 gas.....	Ansted.....	Oct. 7, 1907
11	Wyoming.....	Dietz, Sheridan County ^b	No. 1.....	Sept. 24, 1907
16	Virginia.....	Pocahontas, Tazewell County.	No. 3.....	Baby.....	Nov. 23, 1907
18	Wyoming.....	Diamondville, Uinta County.	No. 1.....	Dec. 9, 1907
23	Illinois.....	Harrisburg, Saline County	No. 5.....	No. 8.....	Jan. 30, 1908
25	Utah.....	Castle Gate, Carbon County	Mar., 1908
46	Wyoming.....	Near Kemmerer, Uinta County.	Willow Creek.....	Mar., 1909

^a Special laboratory number not corresponding to those of the general laboratory of the fuel-testing plant.

^b Subbituminous.

TABLE 4.—Analyses of coals tested.

Laboratory No.....	1.	3.	10.	11.	16.	18.	23.	25.	46.
<i>Bulk sample (100 pounds).</i>									
Air-drying loss.....	1.63	0.10	0.69	14.63	0.81	2.30	5.21	0.56	1.64
<i>Analysis of air-dried sample:</i>									
Moisture.....	7.67	1.10	.87	11.45	.35	2.64	1.96	2.30	2.17
.....	7.40	1.09	.98	10.83	.59	3.48
Volatile matter.....	30.38	30.67	32.46	35.74	20.93	42.23	32.05	40.24	34.01
Fixed carbon.....	54.32	60.35	61.66	47.74	75.51	50.65	56.75	51.38	58.37
Ash.....	7.63	7.88	5.01	5.07	3.21	4.48	9.24	6.08	5.45
<i>Computed to "as received" basis:</i>									
Moisture.....	9.19	1.18	1.55	24.40	1.16	4.88	7.07	2.85	3.74
.....	21.96
Volatile matter.....	29.89	30.64	32.22	30.52	20.76	41.25	30.37	40.04	33.46
Fixed carbon.....	53.41	60.31	61.26	40.75	74.90	49.49	53.80	51.06	57.44
Ash.....	7.51	7.87	4.97	4.33	3.15	4.38	8.76	6.05	5.36
Sulphur.....38	.61	.41	1.48	.42	.91
Nitrogen.....	1.15	1.07	.97	1.27	1.16
<i>1-pound mine sample corresponding to above.</i>									
Can No.....	3,249	3,245	3,625	(c)
<i>Analysis on basis of sample as received:</i>									
Moisture.....	11.82	3.24	2.65	12.28
Volatile matter.....	27.66	27.13	29.69	36.96
Fixed carbon.....	55.10	62.52	63.50	45.88
Ash.....	5.42	7.11	4.16	4.88
Sulphur.....	.46	.95	1.29	.45
Carbon.....	67.87	78.00	60.44
Hydrogen.....	5.44	5.24	5.36
Nitrogen.....	1.34	1.23	1.32
Oxygen.....	19.47	7.47	27.55
Calories.....	6,645	7,733	8,104	5,774
British thermal units.....	11,961	13,919	14,587	10,393

^a Second determinations given were made on same samples twelve to eighteen months later.

^b Determined by modified method; Sommerer, E. E., Jour. Am. Chem. Soc., vol. 28, 1906, p. 1002.

^c Air-dried No. 11, same as above.

TESTS IN IRON RETORT ON 400 GRAMS OF COAL FOR BY-PRODUCTS OF COKING.

SUMMARIES OF TESTS.

In order to determine the adaptability of certain western coals to the production of gas, tar, and ammonia as by-products of coke making or of gas manufacture, one southern Illinois coal, two Wyoming coals, and one Utah coal were tested in the iron retort; and for standards of comparison several tests were made on the well-known Connellsville and Pocahontas coals of the Appalachian region. A few experiments also were carried out with the Connellsville coal for the study of the effect of moisture in the coal on the ammonia yield. The following tables give summaries of the average results; elsewhere in the paper will be found detailed data of the individual tests:

TABLE 5.—*Results of by-product tests.*

Laboratory No	16.	3.	23.	11.	11 (air dried).	25.	46.
Number of tests averaged	2	6	2	4	2	2	2
Coke.....per cent.....	79.1	71.4	63.1	44.7	53.0	58.6	63.9
Tar.....do.....	7.2	11.3	11.9	7.1	5.5	12.3	10.3
Water.....do.....	1.3	4.9	10.7	27.5	19.0	11.8	10.0
Ammonia.....pounds of sulphate per ton.....	12.9	23.8	25.3	27.2	26.7	26.3	26.3
CO ₂per cent.....	.44	.72	1.20	8.14	8.41	3.13	2.13
H ₂ S.....do.....	.07	.25	.46	.08	.11	.24	.30
Gas.....cubic feet per ton.....	9,700	8,140	8,400	7,830	8,170	7,620	7,940
Composition of gas: ^b							
Illuminants.....	1.4	3.2	3.0	2.2	2.6	5.7	5.5
CO.....	3.2	5.1	7.4	19.5	.4	14.9	12.3
CH ₄ , C ₂ H ₆ , etc.....	26.4	27.8	c 26.3	18.1	c .6	27.2	25.4
H.....	67.8	61.0	c 56.8	54.0	c 49.3	47.8	53.1
N.....	1.2	2.9	6.5	6.2	4.1	4.4	3.7
Value of n in C _n H _{2n+2}	(c)	1.27	(c)	1.18	(c)	1.32	1.29
Total volatile products (without moisture).....	19.7	27.4	29.8	33.3	35.5	38.5	32.4
Water of constitution.....	.1	3.7	3.6	5.5	7.5	8.9	6.3
Inert volatile matter ^d7	4.7	5.1	14.0	16.3	12.4	8.8

^a Calculated to dry basis at 0° C. and 760 millimeters pressure, free of air and CO₂.

^b Calculated to CO₂ and O free basis.

^c Hydrogen not determined separately by palladium but calculated from combustion; CH₄ probably high and H low.

^d Sum of CO₂, ammonia, and water of constitution.

The conditions in all the above tests were approximately those described under "Methods."

The western coals give larger ammonia yields under the laboratory conditions than the Connellsville coal. The bituminous coals, No. 25 (Utah) and No. 46 (Wyoming), yield a gas of good quality, nearly twice as high in percentage of illuminants as that from the Connellsville coal. The yield is somewhat smaller than the Connellsville yield, however, when the gas is freed of CO₂. Coal No. 23, from Harrisburg, Ill., gives a somewhat higher yield of gas than the Connellsville coal and the gas is of similar composition. All these coals, except No. 11, produce coke in the laboratory test, but this result can not be relied on even as an indication of industrial coking possibilities.

The subbituminous Wyoming coal, No. 11, yields large quantities of gas very high in CO_2 and CO. Comparative tests on this coal as received and air-dried are shown in the above table. In order to be fairly comparable, however, they must be computed to an equivalent basis, that of the moisture-free material, and the results expressed in percentages of dry coal, as follows:

TABLE 6.—*Tests of coal from Sheridan field, Wyoming (laboratory No. 11), computed to dry basis.*

	Coal dis- tilled as received.	Coal dis- tilled in air-dried state.
Coke.....per cent..	57.3	59.9
Tar.....do.....	9.1	6.2
Water.....do.....	35.3	21.5
Ammonia.....pounds sulphate per ton..	35.0	30.2
CO_2per cent..	10.4	9.6
Gas at 0° C. and 760 mm. pressure, free of CO_2 and air, and dry...cubic feet per ton..	10,020	9,250
Water of constitution.....per cent..	7.1	8.5
Inert volatile matter.....do.....	18.0	18.4

The meaning of these figures is that the absolute quantities of products to be obtained from the coal, either as mined or partly dried, amount to the above-stated percentages of the dry material present. The ammonia yield on the basis of dry material is greater from the moist coal than from the air-dried coal; on the same basis also the CO_2 and CO are slightly greater from the moist coal, but not so much greater as to lead to the conclusion that any considerable amount of the large CO_2 and CO production is to be ascribed to the action of water vapor on carbon.

EFFECT OF MOISTURE IN COAL ON AMMONIA YIELD.

In order to test further the influence of moisture on ammonia yield a few tests were run on Connellsville coal moistened so as to contain 9.6 per cent of water, and on the coal as received, with 1.2 per cent of water, under conditions otherwise as nearly the same as possible.

TABLE 7.—Comparative by-product tests on dry and wet coal.

[Coal No. 3, with 1.2 and 9.6 per cent of moisture.]

	Percentages of coal as used.		Percentages of moisture-free coal.	
	Distilled dry.	Distilled wet.	Distilled dry.	Distilled wet.
Number of tests averaged.....	2	2	2	2
Coke.....per cent..	71.4	66.6	72.3	73.7
Tar.....do..	10.2	9.2	10.3	10.2
Water.....do..	5.9	13.3	6.0	14.7
Ammonia.....pounds of sulphate per ton..	25.0	24.3	25.3	26.9
Gas at 0° C. and 760 millimeters pressure, dry..cubic feet per ton..	7,900	6,640	8,000	7,350
Composition of gas:				
CO ₂	1.8	1.8		
Illuminants.....	2.7	2.8		
CO.....	5.0	5.1		
CH ₄ , C ₂ H ₆ , etc.....	29.1	33.3		
H.....	58.7	55.0		
N.....	2.7	2.0		
Value of n in C _n H _{2n+2}	1.28	1.17		

On the basis of dry material, the ammonia yield seems to be greater again with the wet coal. The gas yield is lower with the wet coal, but in composition the gas is somewhat richer in hydrocarbons.

VARIATION IN GAS COMPOSITION DURING TEST.

The analyses of gas samples taken successively during the progress of the 400-gram by-product tests show the familiar increase of hydrogen and decrease of hydrocarbons as the carbonization proceeds. The detailed analyses are given later in this paper. It is of interest to note that samples taken early in the test (after the lapse of 20 to 25 per cent of the coking time) contain in the high-grade coals 50 to 55 per cent of methane and its homologues and that the proportion of ethane and higher homologues is considerably greater in the early samples. The early sample from Pocahontas coal is not only rich in hydrocarbons but contains also a large percentage of hydrogen (40 to 45 per cent), wherein it differs from the other coal types.

INERT VOLATILE MATTER.

In the tabulated results (Table 5, p. 26) are given percentages of inert volatile matter based on the experimental measurement of water of constitution, CO₂, and ammonia. It will be seen that this inert material varies from 4 per cent of the total volatile in Pocahontas coal to 42 per cent in the subbituminous coal of the Sheridan field, Wyoming. The results of tests on 10-gram samples of coal given on pages 38-40 and 47-54 are similar to those above noted. These values obtained experimentally are somewhat smaller than those obtained by calculation, where all of the oxygen is assumed to combine with hydrogen, forming water.

S. W. Parr ^a proposes a method for the determination of available hydrogen in coal by reference to curves which show the relation of available hydrogen to volatile carbon and total carbon. These curves are plotted from the data of ultimate analyses and from available hydrogen values calculated by converting all of the oxygen to water. The sum of available hydrogen, volatile carbon, and sulphur is taken as combustible volatile and the difference between this and the total volatile as inert volatile. Many of the values thus deduced for the inert volatile matter are higher than those obtained in the present investigation by experimental measurement on account of the presence of CO₂ and CO in the volatile products; but the experimental results of the present investigation uphold the contention of Parr and others that considerable proportions of inert non-combustible materials are contained in the volatile products of coal and that this factor varies with the type of coal.

TESTS IN PLATINUM RETORT ON 10 GRAMS OF COAL.

TOTAL GAS AT VARIOUS TEMPERATURES.

In order to compare different coals in the amount and composition of their total gaseous products at medium and high temperatures, a series of tests was run in the platinum retort on 10 grams of coal, placed in the furnace at certain fixed temperatures and heated until gas practically ceased to be evolved. These tests were run in an atmosphere of nitrogen; the CO₂ and CO found are therefore products of destructive distillation, not of combustion. The gas analyses as given have been calculated to a nitrogen and oxygen free basis, it being assumed that any oxygen present is due to air admitted accidentally during the mixing and transferring of the gas sample. The volumes given under "gas yield" have also been corrected for air in a few instances where oxygen was shown by the analysis. No determinations of tar and water were made in these tests.

TABLE 8.—Total gas yield and composition at different temperatures. ^b

[From 10 grams air-dried coal.]

Temperature of furnace.....° C..	Coal No. 1 (Ziegler, Ill.).						
	500.	600.	700.	800.	900.	1,000.	1,100.
Highest temperature reached in coal.....° C..	390	480	585	685	811	920	1,026
Gas at 25° C.....cubic centimeters..	197	535	980	1,550	2,335	2,700	3,120
Composition of gas:							
CO ₂	23.8	7.6	6.4	3.9	2.5	2.7	1.8
Illuminants.....	6.5	5.0	4.1	3.3	3.2	3.7	4.0
CO.....	16.5	16.1	21.1	16.9	15.2	15.1	16.1
CH ₄ , C ₂ H ₆ , etc.....	49.5	55.0	41.5	34.4	27.8	23.1	19.4
H.....	3.7	16.3	26.9	41.5	51.3	55.4	58.7
Value of n in C _n H _{2n+2}	1.42	1.29	1.21	1.16	1.22	1.18	1.23

^a Jour. Am. Chem. Soc., vol. 29, 1907, p. 585; Bull. Illinois State Geol. Survey, No. 3, 1906.

^b Data given are averages of results of individual tests to be found elsewhere in this paper.

TABLE 8.—*Total gas yield and composition at different temperatures—Continued.*

Temperature of furnace.....° C..	Coal No. 3 (Connellsville, Pa.).						
	500.	600.	700.	800.	900.	1,000.	1,100.
Highest temperature reached in coal.....° C..	390	474	589	705	812	922	1,010
Gas at 25° C.....cubic centimeters..	161	718	1,220	1,723	2,080	2,900	3,530
Composition of gas:							
CO ₂	15.9	4.2	3.2	2.0	1.1	1.2	1.0
Illuminants.....	9.1	7.1	4.3	4.5	4.8	4.6	5.2
CO.....	7.8	6.0	6.3	7.2	7.4	6.4	7.3
CH ₄ , C ₂ H ₆ , etc.....	63.3	64.4	55.8	47.0	33.2	29.0	26.3
H.....	3.9	18.3	30.4	39.3	53.5	58.8	60.2
Value of n in C _n H _{2n+2}	1.69	1.37	1.27	1.21	1.22	1.11	1.15

Temperature of furnace.....° C..	Coal No. 16 (Pocahontas).			Coal No. 11 (Dietz, Wyo.).		
	500.	700.	1,000.	500.	700.	1,000.
Highest temperature reached in coal.....° C..	615	920	600	920
Gas at 25° C.....cubic centimeters..	238	1,185	3,230	517	1,300	3,650
Composition of gas:						
CO ₂	5.5	1.4	.4	54.3	21.7	10.4
Illuminants.....	5.2	3.5	3.7	3.7	3.5	4.5
CO.....	3.5	5.1	4.6	19.6	21.5	22.3
CH ₄ , C ₂ H ₆ , etc.....	70.6	55.2	26.8	18.9	29.4	16.3
H.....	15.2	34.8	64.5	3.5	23.9	46.5
Value of n in C _n H _{2n+2}	1.46	1.19	1.13	1.55	1.19	1.25

TABLE 9.—*Yield of different gaseous products at 500°, 700°, and 1,000° C.*

(Cubic centimeters from 10 grams of coal.)

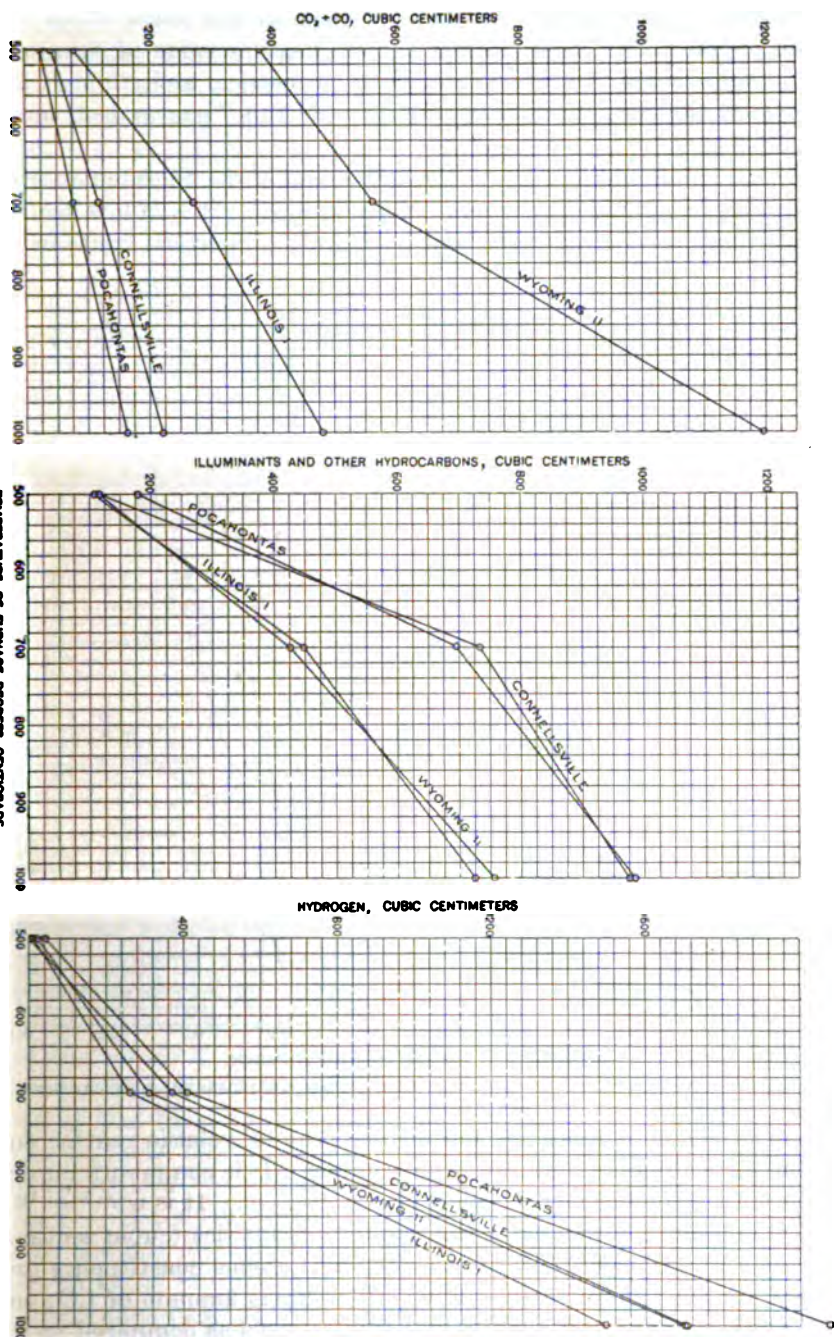
Temperature of furnace.....° C..	500.				700.				1,000.			
	16.	3.	1.	11.	16.	3.	1.	11.	16.	3.	1.	11.
No. of coal.....												
Total gas.....	238	161	197	517	1,185	1,220	980	1,300	3,230	2,900	2,700	3,650
CO ₂	13	26	47	281	17	39	63	282	13	35	73	380
Illuminants.....	12	15	13	19	42	52	40	46	120	133	100	164
CO.....	8	13	33	101	60	77	207	280	149	186	408	814
CH ₄ , C ₂ H ₆ , etc.....	168	102	97	98	654	681	407	382	866	841	624	595
H.....	37	5	7	18	412	371	263	310	2,082	1,705	1,495	1,697
Value of n in C _n H _{2n+2}	1.46	1.69	1.42	1.55	1.19	1.27	1.21	1.19	1.13	1.11	1.18	1.25

The coals used in the tests above reported were selected as being representative of more or less well defined types and localities.

No. 1 is from the Zeigler mine, in the so-called No. 7 seam of Illinois. It is representative of the noncoking Interior Province coals, among which it ranks high in heating value and steaming qualities. It has high volatile matter and gives trouble by the formation of mine gas.

No. 3 is the well-known coking coal of Connellsville, Pa., of higher heating value and slightly lower volatile matter than No. 1. It is fairly representative of the high-grade steaming and coking coals of the Pittsburg district. This type is commonly burned with large smoke production in boiler plants.

FIGURE 3.—Total quantities of different gases from 10 grams of air-dried coal at different temperatures.



No. 16 is the well-known Pocahontas coal, representative of the high-grade, low-volatile, "smokeless" coals of the lower West Virginia region. This type has the highest heating value of American bituminous coals, has excellent coking properties, and, though low in volatile matter, gives a large quantity of "thin" gas on destructive distillation.

No. 11 is the subbituminous, low-grade coal of the Sheridan district, Wyoming, commonly known as "black lignite." It is of low heating value, high oxygen content, and no coking properties; contains 20

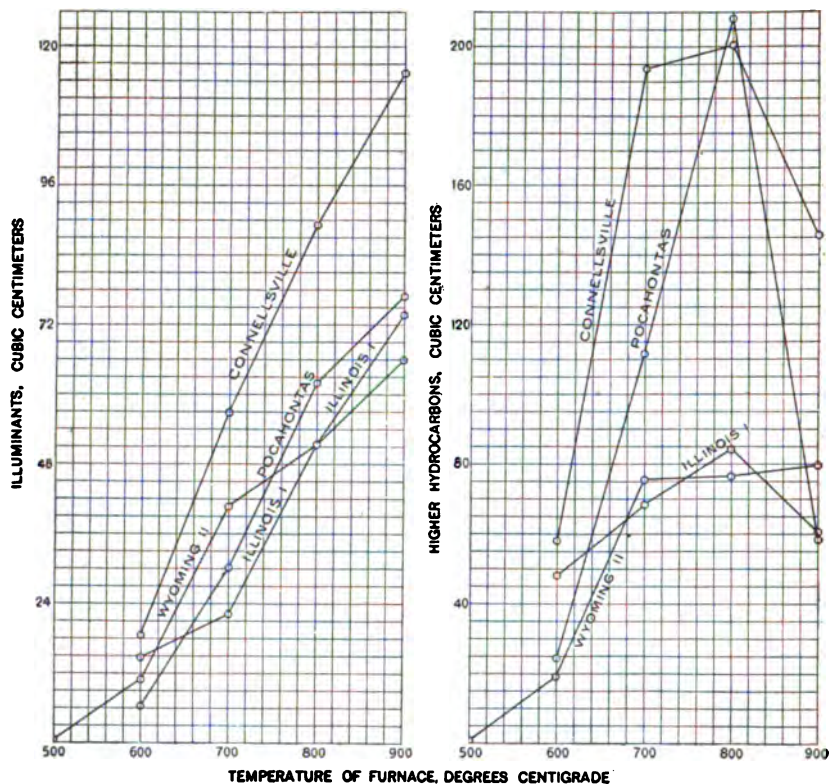


FIGURE 4.—Smoky constituents of early volatile matter; 10 grams of air-dried coal heated ten minutes.

to 24 per cent of moisture and 30 per cent of volatile matter and crumbles badly on exposure to the weather. It is commonly burned with very low efficiency in boiler plants.

Table 9 and figure 3 show the absolute quantities (cubic centimeters per 10 grams of coal) of the different gaseous constituents produced at the several temperatures. The great amount of CO, and CO produced by the Wyoming and Illinois coals as compared to the yield of the eastern coals is very noticeable. The eastern coals, on the other hand, produce considerably greater amounts of hydrocar-

bons and hydrogen even at the low and medium temperatures. Hydrogen is given off more abundantly by the Pocahontas coal than by the others, at both low and high temperatures. As shown by the value of n in the saturated hydrocarbons C_nH_{2n+2} , the proportion of higher members of this hydrocarbon series is greater at low temperatures than at high, and greater in the Connellsville type than in the others. In some tests less than half of these hydrocarbons is methane, the remainder being ethane and higher homologues.

VOLATILE MATTER DURING EARLY PERIOD OF HEATING AT VARIOUS TEMPERATURES.

The character of the volatile matter produced during the early stages of heating has more direct bearing on economy in the utilization of fuel, on smoke production, and on the efficiency of furnaces than that of the volatile matter produced at later stages in the heating. It seems probable that in the first products driven off is involved the chief difficulty in obtaining complete combustion. Accordingly, a series of experiments was undertaken to show comparatively the amount and character of the volatile products which different coals evolve during the first brief period of heating at various temperatures. In one series of experiments the period of heating was fixed by the time element alone, ten minutes being adopted as the period best suited to the amount of coal and method of heating. In another series the rise of temperature in the interior of the coal itself was allowed to fix the period, the heating being stopped as soon as the inside temperature reached a point 200° below that of the furnace. By the latter method the period is, at the different temperatures,

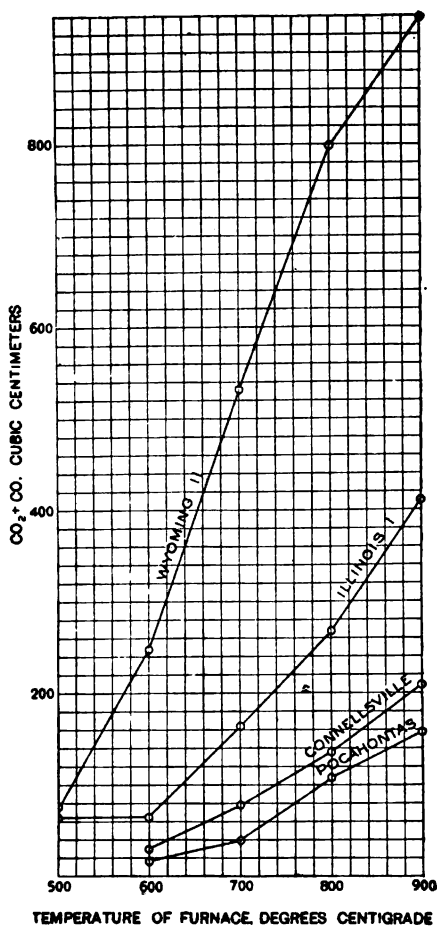


FIGURE 5.—Oxides of carbon in early volatile matter; 10 grams of air-dried coal heated ten minutes.

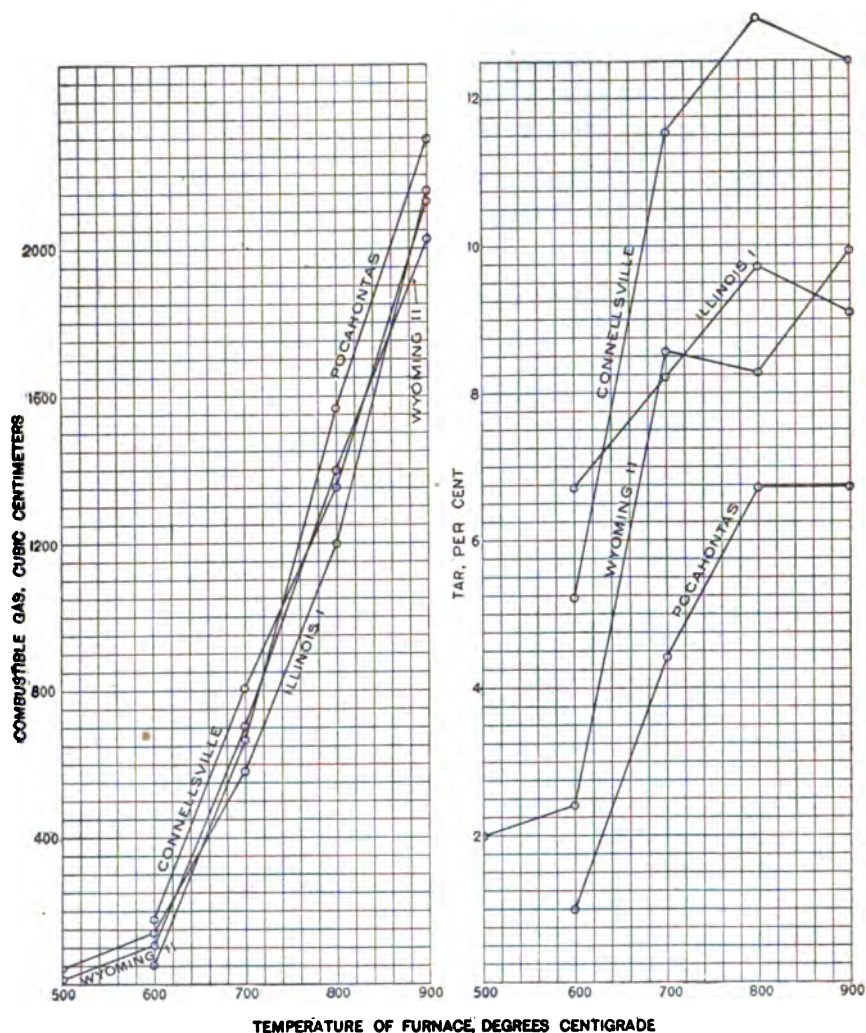


FIGURE 6.—Combustible gases and tar in early volatile matter; 10 grams of air-dried coal heated ten minutes.

more nearly in uniform relation to the total time of evolution of volatile matter.

The coals used for the determination of total gas yield were used also in these experiments; and for the second series two others were added, No. 10, a high-volatile West Virginia gas coal, and No. 18, a high-volatile "long flame" bituminous coal from Diamondville, Wyo. The latter does not coke but is valued locally as a high-grade steam-

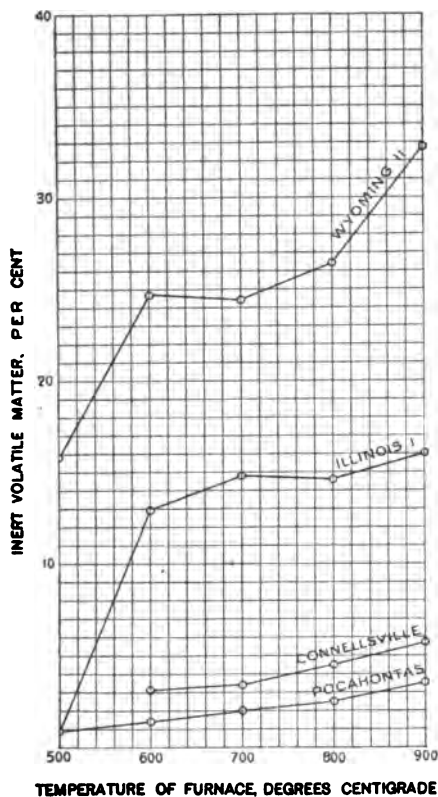


FIGURE 7.—Inert or noncombustible constituents (including moisture) of early volatile matter; 10 grams of air-dried coal heated ten minutes.

ing coal and is particularly well adapted to reverberatory furnaces in smelters, as it gives a long, hot flame. Determinations were made also of tar and water in this early volatile matter.

TABULATED RESULTS, EARLY VOLATILE MATTER.

The results, as compiled from the detailed data given later in the paper, are stated in the following tables. The relation of the composition and volume of the volatile matter to the temperature in the two series of experiments is shown graphically in figures 4 to 9.

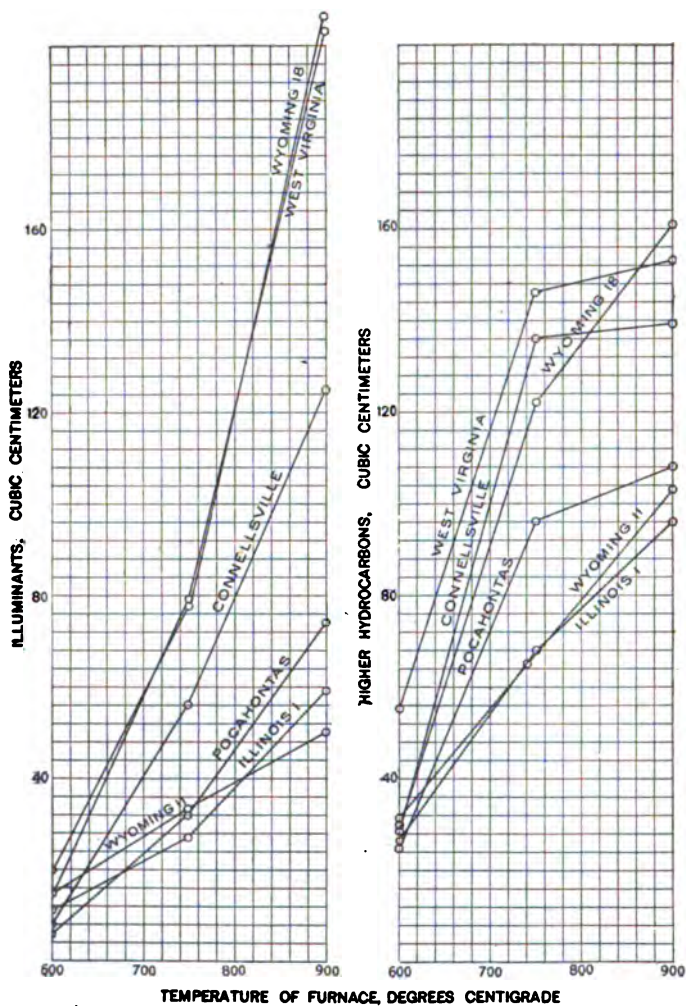


FIGURE 8.—Smoky constituents of early volatile matter; 10 grams of air-dried coal heated to definite temperature.

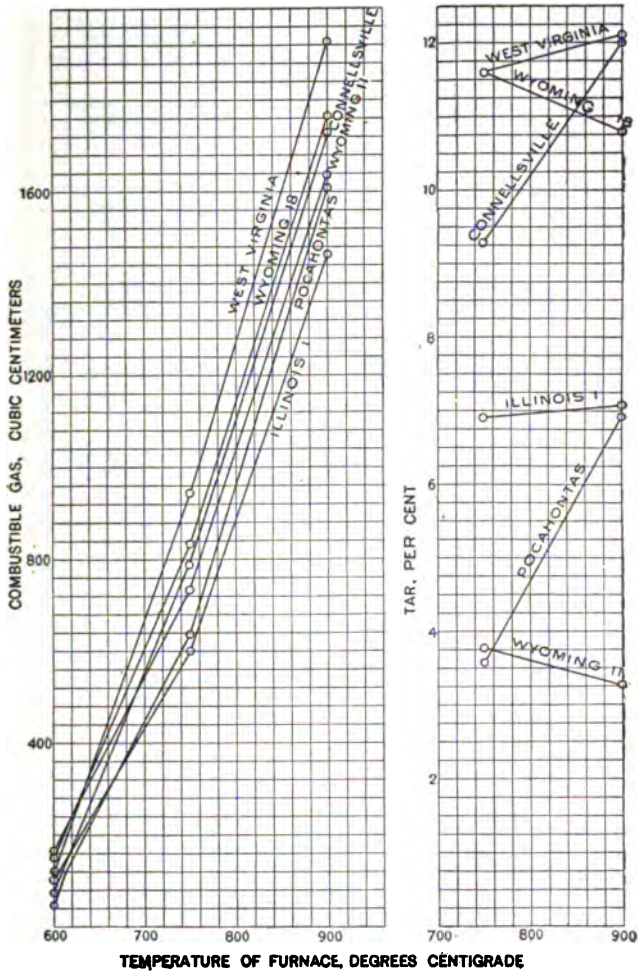


FIGURE 9.—Combustible gases and tar in early volatile matter; 10 grams of air-dried coal heated to definite temperature.

TABLE 10.—Volatile matter in ten minutes' heating of 10 grams of air-dried coal.

	Coal No. 1 (Zeigler, Ill.).					Coal No. 3 (Connellsville, Pa.).				
	500.	600.	700.	800.	900.	500.	600.	700.	800.	900.
Temperature of furnace.....° C.										
Highest temperature reached in coal.....° C.										
Tar.....per cent.	345	440	545	690	91	335	441	562	687	12.5
Water.....do.	6.7	12.2	13.6	13.5	14.4	8	5.2	11.6	13.2	12.5
Gas at 25° C.....cubic centimeters.	90	173	640	1,252	2,240	8	2.9	2.9	4.0	4.7
Composition of gas: a										
CO ₂	52.7	19.4	9.4	4.3	3.7		196	829	1,375	2,180
Hydrogen.....	0	8.6	3.5	4.1	3.3		7.5	2.8	1.7	2.4
CO.....	18.9	18.0	16.4	17.0	14.6		10.0	6.9	6.5	5.3
CH ₄ , C ₂ H ₆ , etc.....	28.4	50.8	46.2	37.3	30.1		72.7	58.4	44.0	35.3
H.....	0	3.2	24.5	37.3	48.3		2.6	25.3	30.6	49.9
Value of n in C _n H _{2n+2}	1.55	1.23	1.73	1.18	1.09		1.42	1.40	1.33	1.19
Coal No. 11 (Dietz, Wyo.).										
Temperature of furnace.....° C.										
Highest temperature reached in coal.....° C.										
Tar.....per cent.	0.3	1.0	599	6.7	6.7	2.0	2.4	580	8.6	9.9
Water.....do.	.8	1.2	1.7	2.1	2.7	14.5	21.0	18.1	18.7	24.3
Gas at 25° C.....cubic centimeters.	4	63	675	1,590	2,337	78	283	1,020	1,780	2,447
Composition of gas: a										
CO ₂		14.7	1.9	1.4	1.8	81.5	62.8	30.9	21.5	17.3
Hydrogen.....		10.1	4.5	3.9	3.3	.5	3.8	4.0	2.9	2.7
CO.....		9.3	3.9	5.4	4.9	15.6	21.7	21.4	23.3	21.0
CH ₄ , C ₂ H ₆ , etc.....		63.1	61.4	40.5	35.0	2.4	11.7	27.4	19.6	19.3
H.....		28.3	28.3	48.8	54.4	.0	.0	16.3	32.7	39.7
Value of n in C _n H _{2n+2}		1.48	1.27	1.32	1.07	1.62	1.44	1.27	1.22	1.17

a Computed to nitrogen and oxygen free basis.

TABLE 11.—Quantities of different gases from ten minutes' heating of 10 grams of air-dried coal.

Temperature of furnace..... ° C.		500.				600.				700.			
No. of coal.....	16.	3.	1.	11.	16.	3.	1.	11.	16.	3.	1.	11.	
CO ₂				64	9	14	34	184	13	23	60	315	
CO.....			47	12	6	14	31	64	26	55	105	218	
Illuminants.....			17	0	4	19	15	11	30	57	22	41	
CH ₄ , C ₂ H ₆ , etc.....			26	0.4	40	138	88	34	415	484	296	280	
H.....			0	2	2	5	6	0	191	210	157	166	
Value of n in C ₁₁ H _{2n+2}			0	1.62	1.48	1.42	1.55	1.44	1.27	1.40	1.23	1.27	

Temperature of furnace..... ° C.		800.				900.			
No. of coal.....	16.	3.	1.	11.	16.	3.	1.	11.	
CO ₂	23	23	54	383	42	53	83	423	
CO.....	86	113	213	415	115	155	327	514	
Illuminants.....	62	89	51	52	77	115	66	66	
CH ₄ , C ₂ H ₆	644	605	467	349	832	770	674	971	
H.....	776	544	467	582	1,271	1,088	1,062	971	
Value of n in C ₁₁ H _{2n+2}	1.32	1.33	1.18	1.22	1.07	1.19	1.09	1.17	

TABLE 12.—*Early volatile products in second series of tests.*

[10 grams of air-dried coal heated to definite temperatures, 200° below that of furnace.]

	Coal No. 1.			Coal No. 3.			Coal No. 16.		
Temperature of furnace. ° C.	600.	750.	900.	600.	750.	900.	600.	750.	900.
Time heated..... minutes	8.5	6.3	4.8	7.7	7.5	6.3	8.2	7.3	5.8
Tar..... per cent	6.9	7.1	9.3	12.0	3.6	6.9
Water..... do	14.1	15.2	4.7	4.7	2.3	2.7
Gas at 25° C., cubic centimeters	150	664	1,568	89	817	1,775	65	653	1,640
Composition of gas: ^a									
CO ₂	28.2	9.5	6.6	15.8	3.5	2.6	13.7	2.5	1.8
Illuminants.....	7.1	4.1	3.7	9.2	6.9	7.0	8.6	4.9	4.5
CO.....	16.5	15.4	15.9	7.1	5.5	7.4	6.2	3.7	4.6
CH ₄ , C ₂ H ₆ , etc.....	45.7	48.4	32.3	60.8	61.4	43.5	63.8	61.5	44.0
H.....	2.5	22.6	41.5	7.1	22.7	39.5	7.7	27.4	45.1
Value of n in C _n H _{2n+3}	1.38	1.21	1.19	1.53	1.27	1.18	1.60	1.24	1.15

	Coal No. 11.			Coal No. 18.			Coal No. 10.		
Temperature of furnace. ° C.	600.	750.	900.	600.	750.	900.	600.	750.	900.
Time heated..... minutes	10.1	6.5	3.7	6.8	5.9	5.0	6.6	6.7	6.0
Tar..... per cent	3.8	3.3	11.6	10.8	11.6	12.1
Water..... do	21.2	22.6	11.2	11.9	4.9	4.9
Gas at 25° C., cubic centimeters	407	1,066	2,015	217	934	1,925	136	985	1,983
Composition of gas: ^a									
CO ₂	59.8	30.5	18.9	28.9	10.9	8.2	12.7	4.1	2.7
Illuminants.....	3.7	3.1	2.5	9.3	8.3	8.6	9.7	8.0	8.2
CO.....	20.3	20.7	23.6	17.5	17.3	18.2	6.2	5.8	7.6
CH ₄ , C ₂ H ₆ , etc.....	14.7	24.4	17.6	40.0	44.9	32.1	61.6	57.0	40.5
H.....	1.5	21.3	37.4	4.3	18.6	32.9	9.8	25.1	41.0
Value of n in C _n H _{2n+3}	1.52	1.25	1.29	1.34	1.29	1.26	1.66	1.26	1.19

^a Computed to oxygen and nitrogen free basis.TABLE 13.—*Quantities of various gases (cubic centimeters) produced in second series of tests.*

TEMPERATURE OF FURNACE, 600°; OF COAL, 400°.

No. of coal.....	16.	3.	10.	1.	18.	11.
Time heated..... minutes	8.2	7.7	6.6	8.5	6.8	10.1
CO ₂	9	14	17	42	63	243
CO.....	4	6	8	25	38	83
Illuminants.....	6	8	13	11	20	15
CH ₄ , C ₂ H ₆ , etc.....	41	54	84	68	87	60
H.....	5	7	14	4	9	6
Value of n in C _n H _{2n+3}	1.60	1.53	1.66	1.38	1.34	1.52

TEMPERATURE OF FURNACE, 750°; OF COAL, 550°.

Time heated..... minutes	7.3	7.5	6.7	6.3	5.9	6.5
CO ₂	16	29	40	63	102	321
CO.....	24	45	57	102	161	219
Illuminants.....	32	56	79	27	78	33
CH ₄ , C ₂ H ₆ , etc.....	402	502	562	322	419	268
H.....	179	185	247	160	174	225
Value of n in C _n H _{2n+3}	1.24	1.27	1.26	1.21	1.29	1.25

TEMPERATURE OF FURNACE, 900°; OF COAL, 700°.

Time heated..... minutes	5.8	6.3	6.0	4.8	5.0	3.7
CO ₂	29	46	54	103	158	381
CO.....	75	131	150	249	350	476
Illuminants.....	74	125	163	59	106	50
CH ₄ , C ₂ H ₆ , etc.....	722	772	803	506	618	355
H.....	740	701	813	651	633	733
Value of n in C _n H _{2n+3}	1.15	1.18	1.19	1.19	1.26	1.29

CONCLUSIONS FROM RESULTS ON EARLY VOLATILE MATTER.

The tabulated and plotted data given above lead to several general conclusions.

The early volatile products from the Illinois and Wyoming coals contain large proportions of inert constituents and of carbon monoxide. At the lower temperatures also they contain more combustible gases than those from the eastern coals. Smoky constituents (shown in figs. 4 and 8) are greatest from the Connellsville coal and the West Virginia and Wyoming gas coals. Higher hydrocarbons, such as ethane, are produced in greatest abundance from the eastern coals; the quantity produced rises to a maximum at about 800°, then rapidly falls on account of decomposition by heat. At furnace temperatures of 500° and 600° these higher hydrocarbons constitute about 50 per cent of the total methane hydrocarbons.

On the other hand, when the heating is continued only to a certain temperature in the coal, as in the second series of tests above, the quantities of higher hydrocarbons increase continuously as a higher furnace temperature is used, for the time of heating grows less.

Coals of the low-volatile bituminous type, represented by Pocahontas coal, produce hydrogen abundantly on heating and also large amounts of methane hydrocarbons.

It is indicated that smokeless combustion is more difficult with coals of the Connellsville and the West Virginia "gas" types than with the southern Illinois and the Wyoming subbituminous types. On the other hand, certain western coals like that of Diamondville, Wyo., produce a rich smoky volatile matter at moderate temperatures in larger quantities than the eastern smoky coals. Coals of this Diamondville type, of which there are representatives in several different localities in the West, show every indication of being valuable for the manufacture of illuminating gas and by-products.

It is evident that the geologically older Appalachian coals, as compared to the younger western coals, contain a larger amount of bitumen or substances which readily liberate methane and ethane hydrocarbons and hydrogen. The western coals, on the other hand, in inverse ratio to their geologic age, produce larger amounts of CO₂, CO, and water. The readiness with which CO₂ is liberated in large amounts even at the lower temperatures (300° to 500°) indicates the presence of compounds having the direct carbon-oxygen linking, such as the complex alcohols, aldehydes, or acids.

VOLATILE MATTER AT ORDINARY TEMPERATURES AND AT 105° C.

In connection with experiments on losses during storage of coal, the writers have noted the accumulation, in sealed bottles containing coal, of considerable amounts of methane and, in comparatively rare

cases, also, of CO_2 . Parr and Wheeler^a have previously noted this accumulation of combustible gas in Illinois coals in sealed bottles, but did not collect or analyze the gas. The present writers hope to publish in another bulletin the results of a series of experiments on the storage of coal, in which will be embodied a large amount of data on the accumulation of gas.

These experiments have shown also a remarkably rapid and long-continued absorption of oxygen by coal in storage, with some variation among coals in respect to the amount and rapidity of the absorption. R. T. Chamberlin in his "Notes on explosive mine gases and dusts,"^b previously cited, demonstrates the continuous formation and accumulation of methane from coal in mines, and the rapid absorption by coal dust of oxygen from the air without forming CO_2 .

A series of experiments on the direct weighing of the products driven off in drying powdered coal for two hours at 105°C . showed slight losses of CO_2 (0.1 to 0.4 per cent) with coals from southern Illinois and the Sheridan field, Wyoming, but only traces of hydrocarbons.

TEMPERATURE IN THE "OFFICIAL VOLATILE-MATTER" DETERMINATION.

One gram of powdered coal was heated for seven minutes in the usual manner prescribed for the "official volatile-matter determination," except that a hole was made in the crucible cover and a thermocouple inserted just under the surface of the coal. With coal No. 1 (Illinois) the maximum temperature (830°C .) was reached in three minutes; with coal No. 3 (Connellsville) the maximum (838°C .) was reached in three minutes and a half.

CALCULATION OF THE HEAT VALUE OF COAL FROM ITS ULTIMATE ANALYSIS.

The experimental results given in the preceding pages show conclusively that in the process of breaking down under the influence of heat the coal substance gives up its oxygen partly in the form of carbon-oxygen compounds and partly as the hydrogen-oxygen compound water. Some recent work by Vignon in Europe supports this conclusion.^c The figures demonstrating this point are given again below in more compact form.

^a Bull. 17, Univ. Illinois Eng. Exper. Sta., 1907, p. 33.

^b Bull. U. S. Geol. Survey No. 383, 1909.

^c Vignon, L., Bull. Soc. chim., 4th ser., vol. 3, p. 109.

TABLE 14.—*Oxygen relations in volatile matter.*

[Values are percentages of air-dried coal.]

Coal.	Per cent of oxygen compounds in volatile matter.			Oxygen in CO and CO ₂ .	Oxygen in water.	Total oxygen in dry coal.
	CO ₂ ^a	CO.	Water of constitution.			
No. 16 (Pocahontas):						
400-gram tests.....	0.44	1.21	0.1	1.01	0.09	3.18
10-gram tests.....	.90	1.74	1.5	1.66	1.33	3.18
No. 3 (Connellsville):						
400-gram tests.....	.72	3.7	3.29	5.23
10-gram tests.....	1.04	2.33	3.5	2.08	3.10	5.23
No. 1 (Zeigler, Ill.):						
10-gram tests.....	1.66	4.90	6.7	4.01	5.95	9.12
No. 11 (Sheridan, Wyo.; air dried):						
400-gram tests.....	8.60	6.90	7.5	10.19	6.67	16.63
10 gram tests.....	8.80	8.10	11.03	16.63

^a There is a possibility of the formation of CO₂ in slight amount from the oxygen of air in contact with the coal at the beginning of a test. On the assumption that 500 cubic centimeters of air is in contact with the coal, there could be formed, if all its oxygen entered into CO₂, only 0.28 gram of CO₂, or 0.07 per cent, on 400 grams of coal.

Dulong's calculation of heat value from the ultimate analysis assumes that all of the oxygen of the coal combines with hydrogen of the coal during combustion, thereby neutralizing, so to speak, the calorific value of an amount of hydrogen equal to $O \div 8$. Dulong's calculation, as is well known, gives for many coals less calories than are shown by experimental determination. The coals exhibiting these discrepancies are usually medium and low grade coals, high in oxygen, which, as has been shown above, give up their oxygen in large part combined with carbon instead of with hydrogen.

By combining with carbon instead of with hydrogen in the coal the oxygen exercises less anticalorific influence on the efficiency of the coal, as 1 gram of oxygen in combining with carbon to CO₂ neutralizes three-eighths of a gram of carbon, or 3,030 calories; in combining with carbon to CO it partly neutralizes three-fourths of a gram of carbon, thus neutralizing $\frac{3}{4} \times 2,417 = 1,813$ calories; whereas 1 gram of oxygen in combining with hydrogen to H₂O neutralizes one-eighth gram of hydrogen, or $\frac{1}{8} \times 34,460 = 4,308$ calories. The anticalorific influence of a unit of oxygen in forming CO₂ or CO is therefore approximately 70 per cent or 42 per cent, respectively, of its influence when forming water.

In a paper before the Geological Society of Washington^a David White called attention to the uniform anticalorific influence which oxygen exercises in coal and based partly on this factor a scheme for the classification of coals. He states that the negative value of 1 per cent of oxygen in "ordinary bituminous coals" is probably 70 to 80 calories, or "about twice as great as has been supposed." With reference

^a Science, vol. 27, 1908, p. 537. See also White, David, The effect of oxygen in coal: Bull. U. S. Geol. Survey No. 382, 1909.

to this statement one should note that Dulong's calculation assigns a negative value to 1 per cent oxygen of one-eighth of 1 per cent of 34,460, or 43 calories, plus the diluent effect of 1 per cent of the coal, or approximately 65 calories, making a total of 108 calories, 50 per cent greater than the value which White finds from experimental results. This excessive anticalorific value for oxygen in Dulong's method the present writers propose to explain in some degree by the distribution of oxygen between carbon and hydrogen as set forth above.

If the heat values of an Illinois coal (No. 19) and of a Wyoming coal (No. 11) are calculated by Dulong's formula and by formulas based on the distribution of oxygen between H_2O , CO , and CO_2 , according to the experimental results herein set forth, the following values are found:

Composition and heat value of Illinois and Wyoming coals.

	Illinois (No. 19). ^a	Wyoming (No. 11). ^b
Composition (water free):		
Ash.....	6.16	5.56
Carbon.....	76.96	68.90
Hydrogen.....	4.49	4.56
Nitrogen.....	1.52	1.51
Sulphur.....	.52	.51
Oxygen.....	10.35	18.96
Calories:		
Dulong's calculation.....	7,322	6,333
Modified calculation.....	7,453	6,524
Determined.....	7,536	6,582

^a From Zeigler, Franklin County, Ill.

^b From Dietz, near Sheridan, Wyo. (subbituminous).

The formulas used for the modified calculation are the following:

Illinois type:

$$\text{Calorific value} = 8,080(C - 0.120 \times O) + 34,460(H - 0.063 \times O) + 2,250S.$$

Wyoming type:

$$\text{Calorific value} = 8,080(C - 0.182 \times O) + 34,460(H - 0.050 \times O) + 2,250S.$$

DETAILED DATA OF INDIVIDUAL TESTS.

Full details of the tests on which the conclusions presented in this bulletin are based are given in the subjoined tables:

TABLE 15.—Results of by-product tests on 400 grams of coal.

Coal No.	Test No.	Coke (per cent).	Tar (per cent).	Water (per cent).	Ammonia (pounds of sulphate per ton).	CO ₂ (per cent).	H ₂ S (per cent).	Gas at 0° C. and 30 inches of mercury (cubic feet per ton).
16	33	79.1	7.2	1.0	13.2	0.44	0.08	10,180
16	34	79.1	7.2	1.5	12.5	.43	.06	9,210
3	39	73.5	-----	-----	25.0	.71	.28	8,650
3	41	70.8	13.7	2.4	22.8	.72	.21	8,670
3	42	70.3	-----	-----	22.0	-----	-----	8,080
3	43	70.8	11.3	5.1	22.8	-----	-----	7,840
3	47	72.2	10.5	5.6	25.5	-----	-----	8,000
3	50	70.5	9.8	6.4	24.4	-----	-----	7,800
a 3	48	67.1	8.5	14.1	24.1	-----	-----	6,960
a 3	49	66.0	9.8	12.5	24.5	-----	-----	6,325
23	31	63.1	12.3	11.0	26.4	1.20	.47	8,400
23	32	63.1	11.5	10.3	24.1	1.19	.44	8,400
11	22	44.8	6.9	-----	28.4	8.10	.08	7,850
11	24	44.4	8.2	26.5	25.5	8.22	.07	7,890
11	25	44.9	6.5	27.0	25.9	-----	-----	-----
11	40	45.0	6.8	29.0	29.1	8.03	.10	7,760
b 11	36	52.9	5.3	19.0	28.2	8.62	.12	8,540
b 11	37	53.1	5.6	19.0	25.1	8.20	.10	7,790
25	51	58.4	12.1	11.5	25.8	3.07	.25	7,640
25	53	58.7	12.4	12.0	26.7	3.20	.22	7,600
46	54	63.3	9.9	10.8	25.5	1.86	.30	7,950
46	57	64.4	10.6	9.2	27.1	1.73	-----	7,930

a Coal moistened to 9.6 per cent of moisture.

b Air-dried coal, 11.5 per cent of moisture.

TABLE 16.—Analyses of samples of gas obtained successively during by-product tests.

The analyses under each test are stated in order in which they were made. Unless otherwise specified CO₂ was removed from gas during test.]

Coal and test Nos.	Composition of gas.							Value of $\frac{n}{n+1}$ in C_nH_{2n+1} .
	CO ₂ .	Illuminants.	O.	CO.	CH ₄ , C ₂ H ₆ , etc.	H.	N.	
Coal 16:								
Test 33.....	0	3.5	0.5	2.8	38.2	42.7	12.3	(a)
	0	1.6	.4	4.0	31.0	62.6	.5	(a)
	0	.5	.0	.8	5.8	87.9	5.0	(a)
	0	0	.2	.8	3.1	88.3	7.6	(a)
Test 34.....	.2	3.4	.4	0	56.6	31.7	.7	(a)
	0	2.0	.2	4.4	33.0	57.5	2.9	(a)
	0	0	.2	1.0	5.9	87.7	5.2	(a)
	0	0	.3	.9	3.7	89.1	6.0	(a)
Coal 3:								
Test 39.....	0	5.1	.5	6.1	43.5	37.6	7.2	1.25
	0	2.4	.3	6.3	19.3	66.7	5.0	1.18
Test 41.....	0	5.7	5.4	4.1				
	0	4.7	.5	5.8	43.4	42.3	2.3	1.19
	0	0	0	3.4	6.8	84.5	5.3	
	.4	0	.2	3.2	4.7	85.4	6.1	
Test 42.....	b 3.4	6.2	1.6	5.2	49.5	22.0	12.1	1.34
	2.1	3.8	.5	0.4	39.0	45.4	2.8	1.18
	.6	.2	.4	3.6	8.3	83.3	3.6	
	0	.4	0	2.4	8.5	84.3	4.4	
Test 43.....	4.2	5.5	0	5.3	51.1	25.8	8.1	1.31
	2.1	4.2	0	6.3	36.3	47.5	3.6	1.22
	.4	.2	0	5.8	12.0	79.3	2.3	
Test 47.....	b 3.3	5.0	.3	5.3	55.4	27.4	3.4	1.26
Test 48.....	3.7	4.6	.8	5.4	51.6	29.9	4.0	1.28
	2.4	3.0	.1	5.4	35.7	51.7	1.7	1.14
	.5	.5	0	2.8	9.5	83.5	3.2	
Test 49.....	b 4.2	5.4	.7	5.1				
	2.9	3.5	.4	6.5	46.4	38.4	1.9	1.21
	1.4	.7	0	4.6	17.0	73.7	2.6	
Coal 23:								
Test 31.....	0	5.1	.8	7.1	34.5	35.5	17.0	(a)
	0	4.5	0	9.4	32.4	50.9	2.8	(a)
	0	0	.1	5.7	7.3	80.4	6.5	(a)
	0	0	0	3.2	3.0	81.9	11.9	(a)
Test 32.....	0	4.4	.5	6.8	57.7	27.1	3.5	(a)
	0	4.2	.6	9.2				(a)
	0	.1	0	7.3	13.8	76.4	2.4	(a)
	0	0	0	4.3	6.2	80.7	9.0	(a)
Coal 11:								
Test 22.....	.9	3.5	.2	12.3	20.4	50.0	12.9	(a)
	5.8	1.6	.3	24.2	9.3	49.5	9.3	(a)
	2.6	0	.4	22.9	6.5	57.4	10.2	(a)
Test 24.....	4.0	3.4	.2	12.1	22.1	51.7	6.5	(a)
	5.4	1.8	.2	21.9	18.4	49.9	2.4	(a)
	.3	.1	.2	27.5	7.1	55.6	9.2	(a)
Test 25.....	7.4	2.6	.3	10.3	20.2	52.5	6.7	(a)
	10.0	2.2	.4	17.9	19.0	47.3	3.2	(a)
	5.6	.2	.2	27.6	5.9	50.9	9.6	(a)
Test 40.....	1.4	4.8	2.2	19.2		29.8		
	5.2	1.6	.3	22.4	12.4	50.3	7.8	
Coal 11 (air dried):								
Test 36.....	11.7	3.5	.5	17.7	37.2	23.4	6.0	(a)
	12.2	2.6	0	19.5	22.4	40.8	2.5	(a)
	2.4	1.0	.4	18.4	14.2	57.1	6.5	(a)
	.1	0	.3	10.2	9.5	68.0	11.9	(a)
Test 37.....	14.0	3.7	.2	16.2	33.5	25.7	6.7	(a)
	11.4	2.9	.3	19.9	23.9	38.3	3.3	(a)
	0	0	0	22.2	12.3	58.0	7.8	(a)
Coal 25:								
Test 51.....	.4	7.8	0	12.6	52.0	21.3	5.9	1.33
	.8	9.2	0	14.8	35.2	36.6	3.4	1.33
	.5	.9	0	13.9	9.2	68.8	6.7	1.34
	.5	.6	.3	7.5	5.7	76.4	9.0	1.71
Coal 46:								
Test 54.....	0	9.1	0	13.4	41.9	30.3	5.3	1.40
	1.5	9.5	.2	13.6	25.7	46.3	3.2	1.33
	0	.4	.2	10.0	5.3	77.8	6.3	1.68
Test 57.....	1.3	7.2	0	13.3	45.2	32.2	.8	1.31
	0	6.3	.4	12.9	26.1	51.4	2.9	1.25
	.2	.5	.2	8.2	6.9	78.2	5.8	1.48

a Not determined, as H was not separately determined.

b CO₂ not removed during test.

TABLE 17.—Analyses of total gas from by-product tests.

Coal No.	Test No.	Composition of gas. ^a					Value of n in C_nH_{2n+2} .
		Illuminants.	CO.	CH ₄ , C ₂ H ₆ , etc.	H.	N.	
16	33	1.8	3.1	26.0	69.0	.1	(b)
16	34	1.0	3.3	26.8	66.5	2.4	(b)
3	39	3.5	5.2	26.4	60.4	4.5	1.23
3	41	3.9	5.0	21.4	66.5	3.3	1.46
3	42	3.3	5.2	29.3	59.9	2.3	1.19
3	43	3.3	4.9	30.1	58.8	2.9	1.18
3	47	2.7	5.2	30.3	58.1	3.7	1.26
3	50	2.7	5.2	29.3	61.0	1.8	1.29
3	48	2.9	5.0	33.3	58.3	.6	1.12
3	49	2.7	5.4	34.7	53.2	4.0	1.22
23	31	3.0	7.3	29.7	66.3	3.7	(b)
23	32	2.9	7.5	22.8	57.3	9.5	(b)
11	22	2.0	19.9	19.0	53.9	5.2	(b)
11	24	1.9	20.0	18.9	53.7	5.5	(b)
11	40	2.6	17.8	18.1	54.0	7.5	1.18
c11	36	2.5	21.2	21.4	51.6	3.3	(b)
c11	37	2.6	21.6	23.7	47.0	5.1	(b)
25	51	5.8	14.0	27.4	47.7	6.1	1.38
25	53	5.6	15.9	27.0	47.6	3.9	1.25
46	54	6.3	12.8	24.7	51.6	4.6	1.32
46	57	4.6	11.9	25.8	53.9	3.8	1.25

^a Reduced by calculation to CO₂ and air-free basis.^b Not determined, as H was not separately determined.^c Air-dried coal.

TABLE 18.—Total gas obtained from 10 grams of air-dried coal.

COAL NO. 1 (ZEIGLER, ILL.).

Temperature of furnace.....°C.	500.		600.		700.	
Test No.....	60	228	71	78	68	81
Highest temperature in coal.....°C.	390	490	470	580	575
Time to reach highest temperature, minutes.....	20	17	20	16	18
Gas at 25° C.....cubic centimeters.....	130	264	570	500	965	990
Composition of gas:						
As collected—						
CO ₂	7.5	10.7	4.8	5.1	4.2	5.0
Illuminants.....	2.0	3.1	3.0	3.4	2.5	3.4
O.....	1.8	.5	.5	.3	.7	.4
CO.....	4.8	8.3	10.7	10.0	17.7	13.0
CH ₄ , C ₂ H ₆ , etc.....	11.3	32.2	37.3	33.2	33.9	26.7
H.....	.0	4.4	8.9	12.0	17.8	21.1
N.....	72.6	40.8	34.8	36.0	23.2	30.4
Computed to N and O free basis—						
CO ₂	29.3	18.2	7.4	8.0	5.5	7.2
Illuminants.....	7.8	5.2	4.6	5.3	3.3	4.9
CO.....	18.8	14.2	16.5	15.7	23.3	18.8
CH ₄ , C ₂ H ₆ , etc.....	44.1	54.9	57.7	52.2	44.5	38.6
H.....	.0	7.5	13.8	18.8	23.4	30.5
Value of n in C_nH_{2n+2}	1.48	1.36	1.21	1.36	1.18	1.23

TABLE 18.—Total gas obtained from 10 grams of air-dried coal—Continued.

COAL NO. 1 (ZEIGLER, ILL.)—Continued.

Temperature of furnace.....°C.	800.		900.		1,000.	1,100.
Test No.....	85	93	88	97	90	99
Highest temperature in coal.....°C.	679	691	797	832	920	1,026
Time to reach highest temperature, minutes.....	12	12	10	8	7	6
Gas at 25° C.....cubic centimeters.....	1,500	1,600	2,290	2,380	2,700	3,120
Composition of gas:						
As collected—						
CO ₂	3.0	3.4	2.0	2.4	2.2	1.6
Illuminants.....	3.0	2.4	3.2	2.4	3.1	3.5
O.....	.4	.2	.3	1.0	1.5	.3
CO.....	14.0	13.9	13.6	13.2	12.5	14.0
CH ₄ , C ₂ H ₆ , etc.....	25.0	32.1	24.0	25.1	19.2	16.9
H.....	35.4	33.3	45.0	45.4	46.0	51.2
N.....	19.2	14.7	11.9	10.5	15.5	12.5
Computed to N and O free basis—						
CO ₂	3.7	4.0	2.3	2.7	2.7	1.8
Illuminants.....	3.7	2.8	3.7	2.7	3.7	4.0
CO.....	17.5	16.3	15.5	14.9	15.1	16.1
CH ₄ , C ₂ H ₆ , etc.....	31.1	37.8	27.3	28.4	23.1	19.4
H.....	44.0	39.1	51.2	51.3	55.4	58.7
Value of n in C _n H _{2n+2}	1.17	1.15	1.26	1.18	1.18	1.23

* Test run in air instead of in nitrogen.

COAL NO. 3 (CONNELLSVILLE, PA.).

Temperature of furnace.....°C.	500.		600.		700.	
Test No.....	61	227	79	84	77	82
Highest temperature in coal.....°C.	390	487	478	577	577	577
Time to reach highest temperature, minutes.....	24	14	15	13	17	17
Gas at 25° C.....cubic centimeters.....	150	172	730	705	1,280	1,160
Composition of gas:						
As collected—						
CO ₂	3.8	4.8	2.8	2.6	2.6	1.8
Illuminants.....	1.8	3.7	4.4	4.6	3.5	3.0
O.....	3.1	.8	.8	.5	.1	.8
CO.....	1.8	2.4	4.6	3.1	4.2	4.8
CH ₄ , C ₂ H ₆ , etc.....	9.5	34.0	43.9	38.8	41.4	42.4
H.....	.0	3.9	13.2	10.6	23.5	22.5
N.....	80.0	50.4	30.3	39.8	24.7	24.7
Computed to N and O free basis—						
CO ₂	22.0	9.9	4.0	4.4	3.8	2.5
Illuminants.....	10.6	7.6	6.3	7.8	4.5	4.1
CO.....	10.6	5.0	6.6	5.3	5.6	6.9
CH ₄ , C ₂ H ₆ , etc.....	56.8	69.7	64.0	64.9	55.1	56.6
H.....	0	7.8	19.1	17.6	31.0	29.9
Value of n in C _n H _{2n+2}	1.87	1.50	1.39	1.35	1.31	1.23

Temperature of furnace.....°C.	800.		900.	1,000.	1,100.
Test No.....	87	94	89	92	106
Highest temperature in coal.....°C.	689	810	810	922	1,006
Time to reach highest temperature, minutes.....	11	9	9	7	5
Gas at 25° C.....cubic centimeters.....	1,685	1,760	2,080	2,900	3,530
Composition of gas:					
As collected—					
CO ₂	1.6	2.0	.8	1.0	1.0
Illuminants.....	3.8	3.6	3.5	4.0	4.6
O.....	.5	.2	3.7	.2	.3
CO.....	6.2	6.4	5.4	5.6	6.6
CH ₄ , C ₂ H ₆ , etc.....	40.9	40.5	24.2	25.9	23.9
H.....	34.2	33.9	39.0	52.9	54.8
N.....	12.8	13.4	23.4	10.4	8.8
Computed to N and O free basis—					
CO ₂	1.7	2.3	1.1	1.2	1.0
Illuminants.....	4.7	4.2	4.8	4.6	5.2
CO.....	7.1	7.4	7.4	6.4	7.3
CH ₄ , C ₂ H ₆ , etc.....	47.1	46.9	33.2	29.0	26.3
H.....	39.4	39.2	53.5	58.8	60.2
Value of n in C _n H _{2n+2}	1.24	1.18	1.22	1.11	1.15

TABLE 18.—Total gas obtained from 10 grams of air-dried coal—Continued.

COALS NO. 16 (POCAHONTAS, VA.) AND NO. 11 (DIETZ, WYO.).

Temperature of furnace.....°C.	Coal No. 16.					Coal No. 11.		
	500.		700.		1,000.	500.	700.	1,000.
Test No.....	229	230	96	100	107	228	101	104
Highest temperature in coal.....°C.			612	617	920		600	920
Time to reach highest temperature, minutes.....			13	14	8		10	6
Gas at 25° C.....cubic centimeters	240	236	1,190	1,175	3,230	517	1,300	3,650
Composition of gas:								
As collected—								
CO ₂	2.8	3.0	1.0	1.2	.4	39.8	17.9	9.4
Illuminants.....	2.8	2.8	2.8	2.8	3.4	2.7	2.9	4.1
O.....	.8	.7	.4	.4	.4	.2	.3	.5
CO.....	2.3	1.9	4.2	4.0	4.2	14.4	17.7	20.2
CH ₄ , C ₂ H ₆ , etc.....		38.4	45.0	44.2	24.5	13.9	24.2	14.7
H.....		8.3	27.5	28.8	59.0	2.5	19.6	42.1
N.....		44.9	19.1	18.6	8.1	26.5	17.4	9.0
Computed to N and O free basis—								
CO ₂		5.5	1.3	1.5	.4	54.3	21.7	10.4
Illuminants.....		5.2	3.5	3.5	3.7	3.7	3.5	4.5
CO.....		3.5	5.2	4.9	4.6	19.6	21.5	22.3
CH ₄ , C ₂ H ₆ , etc.....		70.6	55.9	54.6	26.8	18.9	29.4	16.3
H.....		15.2	34.1	35.5	64.5	3.5	23.9	46.5
Value of n in C _n H _{2n+1}		1.46	1.17	1.21	1.13	1.55	1.19	1.25

TABLE 19.—Early volatile products from ten minutes' heating of 10 grams of air-dried coal.

COAL NO. 1 (ZEIGLER, ILL.).

Temperature of furnace.....°C.	500.		600.		700.		800.		900.
Test No. a	59	69	73	62	231	76	83		132
Highest temperature in coal.....°C.	345	445	435	550	540	670	689		
Tar.....per cent		6.3	7.1	7.9	8.5	9.7			9.1
Water.....do		11.8	12.7	13.9	13.3	13.5			14.4
Gas at 25° C.....cubic centimeters	90	175	171	485	800	1,224	1,280		2,240
Composition of gas:									
As collected—									
CO ₂	5.0	7.8	7.7	6.4	7.5	3.2	3.4		3.3
Illuminants.....	0	3.4	3.5	3.9	2.8	3.2	3.2		2.9
O.....	2.6	.9	.3	.9	.3	.2	.3		.6
CO.....	1.8	6.3	8.0	10.4	13.1	14.6	13.3		13.0
CH ₄ , C ₂ H ₆ , etc.....	2.7	22.6	18.3		36.9		29.6		26.7
H.....	0	2.3	.4		19.6		29.5		42.9
N.....	87.9	56.7	61.8		19.8		20.8		10.6
Computed to O and N free basis—									
CO ₂	52.7	18.5	20.3		9.4		4.3		3.7
Illuminants.....	0	8.0	9.2		3.5		4.1		3.3
CO.....	18.9	14.9	21.1		16.4		17.0		14.0
CH ₄ , C ₂ H ₆ , etc.....	28.4	53.3	48.3		46.2		37.3		30.1
H.....	0	5.3	1.1		24.6		37.3		48.3
Value of n in C _n H _{2n+1}		1.48	1.62		1.23		1.18		1.09

a Number of test applies to test for gas only.

TABLE 19.—Early volatile products from ten minutes' heating of 10 grams of air-dried coal—Continued.

COAL NO. 3 (CONNELLSVILLE, PA.).

Temperature of furnace.....°C.	500.	600.			700.
Test No.....	66	70	72	75	84
Highest temperature in coal.....°C.	335	455	452	415	565
Tar.....per cent.	5.1	5.3	12.6
Water.....do.	3.1	2.7	3.2
Gas at 25° C.....cubic centimeters	8	162	233	176	668
Composition of gas:					
As collected—					
CO ₂	3.2	3.0	3.2	2.6
Illuminants.....	4.2	4.0	4.3	5.8
O.....5	.9	.6	.3
CO.....	2.9	2.9	3.2	3.7
CH ₄ , C ₂ H ₆ , etc.....	23.6	39.1	30.8
H.....	2.6	.3	.0
N.....	63.0	49.8	57.9
Computed to O and N free basis—					
CO ₂	8.8	6.1	7.7
Illuminants.....	11.5	8.1	10.4
CO.....	8.0	5.9	7.7
CH ₄ , C ₂ H ₆ , etc.....	64.6	79.3	74.2
H.....	7.1	.6	0.0
Value of n in C _n H _{2n+2}	1.65	1.20	1.40

Temperature of furnace.....°C.	700.	800.		900.
Test No.....	65	232	80	133
Highest temperature in coal.....°C.	555	565	695	679
Tar.....per cent.	10.6	12.9	13.5
Water.....do.	2.7	4.2	3.8
Gas at 25° C.....cubic centimeters	990	1,400	1,350
Composition of gas:				
As collected—				
CO ₂	1.7	2.3	1.6	1.0
Illuminants.....	4.7	5.0	5.3	4.5
O.....	3.0	.3	.7	.3
CO.....	4.4	4.7	6.4	5.9
CH ₄ , C ₂ H ₆ , etc.....	35.4	47.3	36.0	30.1
H.....	13.5	22.8	27.6	31.7
N.....	37.3	17.6	22.4	26.5
Computed to O and N free basis—				
CO ₂	2.8	2.8	2.1	1.4
Illuminants.....	7.8	6.1	6.9	6.1
CO.....	7.4	5.7	8.3	8.1
CH ₄ , C ₂ H ₆ , etc.....	59.2	57.6	48.8	41.2
H.....	22.8	27.8	35.9	43.2
Value of n in C _n H _{2n+2}	1.50	1.31	1.25	1.41

COAL NO. 16 (POCAHONTAS, VA.).

Temperature of furnace.....°C.	500.	600.		700.	800.	900.	
Test No.....	124	119	120	122	103	115	127
Highest temperature in coal.....°C.	599
Tar.....per cent.	0.3	0.9	1.0	4.4	6.7	6.7
Water.....do.	.8	1.2	1.3	1.7	2.1	2.8
Gas at 25° C.....cubic centimeters	4	96	40	62	675	1,560	2,325
Composition of gas:							
As collected—							
CO ₂	3.6	2.8	3.0	1.4	1.1	1.2
Illuminants.....	2.8	1.6	2.2	3.2	3.1	3.2
O.....3	1.6	.5	1.6	.3	.2
CO.....	2.2	1.8	1.9	2.8	4.3	4.6
CH ₄ , C ₂ H ₆ , etc.....	14.7	11.1	14.6	44.1	32.6	32.3
H.....	2.2	0	0	20.3	39.2	49.4
N.....	74.2	81.1	77.8	26.6	19.4	9.1
Computed to O and N free basis—							
CO ₂	14.1	16.2	13.8	1.9	1.4	1.8
Illuminants.....	11.0	9.2	10.1	4.5	3.9	3.3
CO.....	8.6	10.4	8.8	3.9	5.4	4.9
CH ₄ , C ₂ H ₆ , etc.....	57.7	64.2	67.3	61.4	40.5	35.6
H.....	8.6	0.0	0.0	28.3	48.8	54.4
Value of n in C _n H _{2n+2}	1.63	1.48	1.32	1.27	1.32	1.07

TABLE 19.—*Early volatile products from ten minutes' heating of 10 grams of air-dried coal—Continued.*

COAL NO. 11 (DIETZ, WYO.).

Temperature of furnace.....°C.	500.		600.		
Test No.....	121	123	116	117	118
Highest temperature in coal.....°C.					
Tar.....per cent.		2.0	3.0	1.8	
Water.....do.		14.5	22.0	20.0	
Gas at 25° C.....cubic centimeters.	94	63	350	270	260
Composition of gas:					
As collected—					
CO ₂	29.4	23.9	34.0	36.2	36.5
Illuminants.....	.4	0	2.2	2.1	2.2
O.....	.8	.5	.4	.4	.7
CO.....	5.6	4.6	14.0	11.6	11.5
CH ₄ , C ₂ H ₆ , etc.....	.4	1.1	9.8	5.6	4.7
H.....	0	0	0	0	0
N.....	63.4	69.9	39.6	44.1	44.4
Computed to O and N free basis—					
CO ₂	82.2	80.7	56.7	65.2	66.5
Illuminants.....	1.1	0	3.7	3.8	4.0
CO.....	15.6	15.6	23.3	20.9	20.9
CH ₄ , C ₂ H ₆ , etc.....	1.1	3.7	16.3	10.1	8.6
H.....	0	0	0	0	0
Value of n in C _n H _{2n+2}	1.25	2.00	1.22	1.48	1.62

Temperature of furnace.....°C.	700.	800.	900.		
Test No.....	105	114	120	128	130
Highest temperature in coal.....°C.	590				
Tar.....per cent.	8.6	8.3	9.8	10.0	
Water.....do.	18.1	18.7	24.8	23.9	
Gas at 25° C.....cubic centimeters.	1,020	1,780	2,400	2,350	2,590
Composition of gas:					
As collected—					
CO ₂	24.8	18.0	14.4	15.7	15.4
Illuminants.....	3.2	2.4	2.4	1.2	2.4
O.....	.2	.2	.3	.5	.2
CO.....	17.2	19.5	20.7	21.8	18.7
CH ₄ , C ₂ H ₆ , etc.....	22.0	16.4			17.1
H.....	13.1	27.3			35.3
N.....	19.5	16.2			10.9
Computed to O and N free basis—					
CO ₂	30.9	21.5			17.3
Illuminants.....	4.0	2.9			2.7
CO.....	21.4	23.3			21.0
CH ₄ , C ₂ H ₆ , etc.....	27.4	19.6			19.3
H.....	16.3	32.7			39.7
Value of n in C _n H _{2n+2}	1.27	1.22			1.17

TABLE 20.—*Early volatile products in second series of tests (heating 10 grams of air-dried coal to definite temperatures).*

COAL NO. 1 (ZEIGLER, ILL.).

Temperature of furnace.....°C..	600.		750.			900.		
Test No.....	155	157	166	167	168	174	175	189
Highest temperature in coal.....°C..	400	400	550	550	550	700	700	700
Time to reach highest temperature.....minutes..	8.7	8.3	7.0	6.0	7.0	5.0	5.0	4.5
Tar.....per cent.	(a)		6.6	7.2		7.0	7.2	
Water.....do.	(a)		14.3	13.9		15.7	14.6	
Gas at 25° C.....cubic centimeters..	149	152	632	696	663	1,490	1,545	1,080
Composition of gas:								
As collected—								
CO ₂	12.9	12.8	6.8	7.3	8.4	6.0	6.2	5.5
Illuminants.....	3.1	3.4	3.0	3.3	3.3	2.9	3.3	3.7
O.....	.5	.4	.2	.4	0	.5	.2	0
CO.....	7.1	7.9	11.9	12.3	11.9	14.6	13.8	14.3
CH ₄ , C ₂ H ₆ , etc.....	20.8	20.9	38.3	37.4	38.0	30.0	29.2	27.8
H.....	.3	2.1	17.9	17.9	16.7	39.1	34.7	37.7
N.....	55.3	52.5	21.9	21.4	21.3	6.9	12.6	11.0
Computed to O and N free basis—								
CO ₂	29.0	27.3	8.7	9.3	10.6	6.5	7.1	6.2
Illuminants.....	7.0	7.2	3.8	4.2	4.2	3.1	3.8	4.2
CO.....	16.1	16.9	15.3	15.8	15.1	15.8	15.9	16.1
CH ₄ , C ₂ H ₆ , etc.....	47.3	44.1	49.1	47.9	48.3	32.4	33.4	31.2
H.....	.6	4.6	23.1	22.8	21.8	42.2	39.8	42.3
Value of n in C _n H _{2n+1}	1.33	1.43	1.21	1.20	1.22	1.19	1.19	1.18

a Tar and water determinations not made at 600° in this series.

COAL NO. 3 (CONNELLSVILLE, PA.).

Temperature of furnace.....°C..	600.			750.			900.	
Test No.....	150	151	152	163	164	165	172	173
Highest temperature in coal.....°C..	400	400	400	550	550	550	700	700
Time to reach highest temperature.....minutes..	7.0	8.7	7.5	8.0	7.0	7.5	6.3	6.3
Tar.....per cent.				9.0	9.9	9.0	12.0	12.0
Water.....do.				5.0	4.3	4.8	4.5	4.9
Gas at 25° C.....cubic centimeters..	88	79	90	775	847	830	1,740	1,810
Composition of gas:								
As collected—								
CO ₂	5.4	5.8	4.9	3.3	2.4	2.7	2.1	2.5
Illuminants.....	3.6	2.8	3.0	5.5	5.5	5.5	6.2	6.4
O.....	0	.3	.6	.5	.1	.4	0	0
CO.....	2.1	2.2	2.9	4.6	4.6	4.1	6.2	7.1
CH ₄ , C ₂ H ₆ , etc.....	21.3	20.4	20.3	47.7	50.4	49.8	40.6	38.5
H.....	4.4	1.4	1.7	17.1	18.8	18.6	35.6	36.2
N.....	63.2	67.1	66.6	21.3	18.2	18.9	9.3	9.3
Computed to O and N free basis—								
CO ₂	14.6	17.8	14.9	4.1	3.0	3.3	2.3	2.8
Illuminants.....	9.9	8.5	9.1	7.1	6.8	6.8	6.9	7.1
CO.....	5.8	6.7	8.8	5.9	5.4	5.1	6.9	7.8
CH ₄ , C ₂ H ₆ , etc.....	57.8	62.7	62.0	60.1	62.2	61.8	44.7	42.4
H.....	11.9	4.3	5.2	22.8	22.6	23.0	39.2	39.9
Value of n in C _n H _{2n+1}	1.63	1.42	1.54	1.27	1.26	1.29	1.18	1.19

TABLE 20.—*Early volatile products in second series of tests (heating 10 grams of air-dried coal to definite temperatures)*—Continued.

COAL NO. 16 (POCAHONTAS).

Temperature of furnace.....°C..	600.			750.		900.	
Test No.....	153	156	159	161	162	176	177
Highest temperature in coal.....°C.	400	400	400	550	550	700	700
Time to reach highest temperature.....minutes	8.0	8.0	8.5	7.3	7.3	5.8	5.8
Tar.....per cent.	3.7	3.6	6.6	7.1
Water.....do.	2.2	2.3	2.9	2.5
Gas at 25° C.....cubic centimeters..	67	62	77	647	659	1,650	1,630
Composition of gas:							
As collected—							
CO ₂	3.4	2.6	4.0	1.9	1.8	1.6	1.5
Hydrocarbons.....	2.0	2.2	2.1	3.7	3.9	4.1	4.0
O.....	.3	.3	.8	.6	.4	.5	0
CO.....	1.8	1.1	1.6	3.0	2.7	3.7	4.5
CH ₄ , C ₂ H ₆ , etc.....	16.1	13.5	16.5	48.2	46.8	39.2	39.2
H.....	1.0	3.4	1.1	21.2	21.2	39.8	40.5
N.....	75.4	76.9	73.9	21.4	23.2	11.1	10.3
Computed to O and N free basis—							
CO ₂	13.8	11.4	15.8	2.5	2.4	1.8	1.7
Hydrocarbons.....	8.1	9.5	8.2	4.8	5.1	4.6	4.4
CO.....	7.3	5.0	6.3	3.9	3.5	4.2	5.0
CH ₄ , C ₂ H ₆ , etc.....	66.7	59.2	65.4	61.8	61.2	44.3	43.7
H.....	4.1	14.9	4.3	27.0	27.8	45.1	45.2
Value of n in C _n H _{2n+2}	1.54	1.71	1.55	1.21	1.27	1.15	1.15

COAL NO. 11 (DIETZ, WYO.).

Temperature of furnace.....°C..	600.		750.			900.*		
Test No.....	158	160	169	170	171	178	179	182
Highest temperature in coal.....°C.	400	400	550	550	550	700	700	700
Time to reach highest temperature.....minutes	10.2	10.0	7.0	6.0	6.5	3.7	3.7	3.7
Tar.....per cent.	3.8	3.8	2.9	3.7
Water.....do.	21.0	21.3	23.1	22.1
Gas at 25° C.....cubic centimeters..	408	406	1,048	1,018	1,100	1,990	2,050	2,000
Composition of gas:								
As collected—								
CO ₂	42.0	41.6	25.6	25.8	25.6	17.5	16.9	17.0
Hydrocarbons.....	2.8	2.4	2.6	2.6	2.5	2.3	2.1	2.3
O.....	.1	.4	.4	.3	.2	.2	.4	.2
CO.....	14.5	14.0	17.3	17.7	17.3	21.5	21.4	21.2
CH ₄ , C ₂ H ₆ , etc.....	10.7	10.1	21.5	20.3	19.7	17.5	14.7	15.8
H.....	1.2	.9	17.4	18.0	18.2	33.7	34.5	33.4
N.....	28.7	30.6	15.2	15.3	16.5	7.3	10.0	10.1
Computed to O and N free basis—								
CO ₂	59.2	60.3	30.3	30.5	30.7	18.9	18.9	18.9
Hydrocarbons.....	3.9	3.5	3.1	3.1	3.1	2.5	2.3	2.6
CO.....	20.4	20.3	20.5	20.9	20.8	23.2	23.9	23.6
CH ₄ , C ₂ H ₆ , etc.....	14.8	14.6	25.5	24.0	23.6	18.9	16.4	17.6
H.....	1.7	1.3	20.6	21.5	21.8	36.5	38.5	37.3
Value of n in C _n H _{2n+2}	1.48	1.56	1.22	1.25	1.29	1.21	1.39	1.28

* Tests on coal No. 11 at 900° were run on coarsely-powdered material (between 10 and 20 mesh) in order to avoid mechanical loss.

TABLE 20.—*Early volatile products in second series of tests (heating 10 grams of air-dried coal to definite temperatures)*—Continued.

COAL NO. 18 (DIAMONDVILLE, WYO.)

Temperature of furnace.....°C.	600.		750.		900.	
Test No.....	225	225a	185	186	180	181
Highest temperature in coal.....°C.	400	400	550	550	700	700
Time to reach highest temperature.....minutes.	6.5	7.0	5.5	6.2	5.0	5.0
Tar.....per cent.	11.6	11.6	11.6	11.6	11.2	10.4
Water.....do.	11.3	11.3	11.3	11.0	11.6	12.2
Gas at 25°C.....cubic centimeters.	228	206	943	926	1,900	1,950
Composition of gas:						
As collected—						
CO ₂	14.7	15.7	9.5	8.7	7.5	7.2
Illuminants.....	5.1	4.7	7.2	6.5	7.7	8.0
O.....	.4	.8	.3	.6	.1	.1
CO.....	9.8	8.6	14.3	14.5	16.7	16.1
CH ₄ , C ₂ H ₆ , etc.....	22.9	19.3	37.9	36.6	29.2	28.5
H.....	2.0	2.6	14.9	16.0	28.9	30.5
N.....	45.1	48.3	15.9	17.1	9.9	9.6
Computed to O and N free basis—						
CO ₂	27.0	30.8	11.3	10.5	8.3	8.0
Illuminants.....	9.4	9.2	8.6	7.9	8.5	8.8
CO.....	18.0	16.9	17.1	17.6	18.6	17.8
CH ₄ , C ₂ H ₆ , etc.....	42.0	37.9	45.2	44.5	32.5	31.6
H.....	3.6	5.2	17.8	19.5	32.1	33.8
Value of n in C _n H _{2n+2}	1.55	1.23	1.26	1.31	1.23	1.28

COAL NO. 10 (PAGE, W. VA.)

Temperature of furnace.....°C.	600.		750.		900.	
Test No.....	224	224a	187	188	190	183
Highest temperature in coal.....°C.	400	400	550	550	550	700
Time to reach highest temperature.....minutes.	6.7	6.5	7.0	6.5	6.7	6.0
Tar.....per cent.	11.5	12.8	11.5	12.8	10.4	12.0
Water.....do.	5.1	4.7	5.1	4.7	4.8	4.9
Gas at 25°C.....cubic centimeters.	136	136	975	995	995	2,005
Composition of gas:						
As collected—						
CO ₂	4.6	5.4	3.4	2.9	3.4	2.4
Illuminants.....	3.8	3.8	6.9	5.7	6.3	7.6
O.....	1.0	.7	.2	3.5	.4	.1
CO.....	2.7	2.2	4.8	4.0	5.1	7.1
CH ₄ , C ₂ H ₆ , etc.....	26.3	22.1	47.5	40.0	48.6	35.6
H.....	2.2	5.7	20.7	18.2	21.0	37.7
N.....	59.4	60.1	16.5	25.7	15.2	9.5
Computed to O and N free basis—						
CO ₂	11.6	13.8	4.1	4.1	4.0	2.7
Illuminants.....	9.6	9.8	8.3	8.1	7.5	8.4
CO.....	6.8	5.6	5.8	5.6	6.0	7.9
CH ₄ , C ₂ H ₆ , etc.....	66.4	56.7	57.0	56.5	57.6	39.3
H.....	5.6	14.1	24.8	25.7	24.9	41.7
Value of n in C _n H _{2n+2}	1.53	1.79	1.28	1.25	1.24	1.22

GENERAL SUMMARY.

The investigations described in this bulletin contribute data on the composition of the volatile products from different kinds of coal as evolved at different temperatures. The comparatively large amounts of inert constituents such as CO₂ and water in the products from certain western coals; the large amounts of higher methane hydrocarbons, such as ethane, in the products at moderate temperatures, particularly from the Appalachian coals; and the larger amounts of gas and tarry vapors produced quickly at moderate temperatures from the younger western coals are the main features of the results. The bearing of these results on smoke-producing

tendencies, on studies of the nature of coal substance, and on the calculation of heat value from ultimate analysis is brought out in the foregoing pages. It has been shown that certain bituminous coals of the West are well adapted to the manufacture of a high-grade illuminating gas and of other by-products of coking, notably ammonia.

Any statement as to the character of the gases or volatile products evolved from coal at specified temperatures has little value unless it is accompanied by a clear description of the conditions prevailing and particularly of the points at which temperatures were taken and of the mass of coal which was heated. The temperature varies throughout the mass and is affected by the rate and time of heating. Temperatures outside of the containing vessel produce different temperatures in the coal itself according to the kind of vessel and the time of heating. The distance of the vessel from the point where temperatures are read influences the difference between the observed temperature and that of the coal within the vessel.

It is expected that these investigations will be continued and that the laboratory results will be correlated with experiments on furnaces and gas producers in operation. The work will include the examination of the composition of gases at various points and the study of losses through incomplete combustion. Further laboratory work will be undertaken for the study of the effect of heating the same coal to certain temperatures at different rates, and it is planned to extend the investigation to a greater variety of coals.

PUBLICATIONS ON FUEL TESTING.

The following publications, except those to which a price is affixed, can be obtained free by applying to the Director of the Bureau of Mines, Washington, D. C. The priced publications can be purchased from the Superintendent of Documents, Government Printing Office, Washington, D. C.:

PUBLICATIONS OF THE UNITED STATES GEOLOGICAL SURVEY.

BULLETIN 261. Preliminary report on the operations of the coal-testing plant of the United States Geological Survey at the Louisiana Purchase Exposition, in St. Louis, Mo., 1904; E. W. Parker, J. A. Holmes, M. R. Campbell, committee in charge. 1905. 172 pp. 10 cents.

PROFESSIONAL PAPER 48. Report on the operations of the coal-testing plant of the United States Geological Survey at the Louisiana Purchase Exposition, St. Louis, Mo., 1904; E. W. Parker, J. A. Holmes, M. R. Campbell, committee in charge. 1906. In three parts. 1492 pp., 13 pls. \$1.50.

BULLETIN 290. Preliminary report on the operations of the fuel-testing plant of the United States Geological Survey at St. Louis, Mo. 1905, by J. A. Holmes. 1906. 240 pp. 20 cents.

BULLETIN 323. Experimental work conducted in the chemical laboratory of the United States fuel-testing plant at St. Louis, Mo., January 1, 1905, to July 31, 1906, by N. W. Lord. 1907. 49 pp. 10 cents.

BULLETIN 325. A study of four hundred steaming tests made at the fuel-testing plant, St. Louis, Mo., 1904, 1905, and 1906, by L. P. Breckenridge. 1907. 196 pp. 20 cents.

BULLETIN 332. Report of the United States fuel-testing plant at St. Louis, Mo., January 1, 1906, to June 30, 1907; J. A. Holmes, in charge. 1908. 299 pp. 25 cents.

BULLETIN 334. The burning of coal without smoke in boiler plants; a preliminary report, by D. T. Randall. 1908. 26 pp. 5 cents.

BULLETIN 336. Washing and coking tests of coal and cupola tests of coke, by Richard Moldenke, A. W. Belden, and G. R. Delamater. 1908. 76 pp.

BULLETIN 339. The purchase of coal under government and commercial specifications on the basis of its heating value, with analyses of coal delivered under government contracts, by D. T. Randall. 1908. 27 pp. 5 cents.

BULLETIN 343. Binders for coal briquets, by J. E. Mills. 1908. 56 pp.

BULLETIN 362. Mine sampling and chemical analyses of coals tested at the United States fuel-testing plant, Norfolk, Va., in 1907, by J. S. Burrows. 1908. 23 pp.

BULLETIN 363. Comparative tests of run-of-mine and briquetted coal on locomotives, including torpedo-boat tests and some foreign specifications for briquetted fuel, by W. F. M. Goss. 1908. 57 pp., 4 pls.

BULLETIN 366. Tests of coal and briquets as fuel for house-heating boilers, by D. T. Randall. 1908. 44 pp., 3 pls.

BULLETIN 367. Significance of drafts in steam-boiler practice, by W. T. Ray and Henry Kreisinger. 1909. 61 pp.

BULLETIN 368. Washing and coking tests of coal at Denver, Colo., by A. W. Belden, G. R. Delamater, and J. W. Groves. 1909. 54 pp., 2 pls.

BULLETIN 373. The smokeless combustion of coal in boiler plants, by D. T. Randall and H. W. Weeks. 1909. 188 pp. 20 cents.

BULLETIN 378. The purchase of coal under government specifications, by J. S. Burrows. 1909. 44 pp. 10 cents.

BULLETIN 382. The effect of oxygen in coal, by David White. 1909. 78 pp., 3 pls.

BULLETIN 385. Briquetting tests at the United States fuel-testing plant, Norfolk, Va., 1907-8, by C. L. Wright. 1909. 41 pp., 9 pls.

BULLETIN 392. Commercial deductions from comparisons of gasoline and alcohol tests on internal-combustion engines, by R. M. Strong. 1909. 38 pp.

BULLETIN 393. Incidental problems in gas-producer tests, by R. H. Fernald, C. D. Smith, J. K. Clement, and H. A. Grine. 1909. 29 pp.

BULLETIN 402. The utilization of fuel in locomotive practice, by W. F. M. Goss. 1909. 28 pp.

BULLETIN 403. Comparative tests of run-of-mine and briquetted coal on the torpedo boat *Biddle*, by Walter T. Ray and Henry Kreisinger. 1909. 49 pp.

BULLETIN 412. Tests of run-of-mine and briquetted coal in a locomotive boiler, by Walter T. Ray and Henry Kreisinger. 1909. 32 pp.

BULLETIN 416. Recent development of the producer-gas power plant in the United States, by R. H. Fernald. 1909. 82 pp., 2 pls.

BULLETIN 428. The purchase of coal by the Government under specifications, with analyses of coal delivered for the fiscal year 1908-9, by G. S. Pope. 80 pp.

PUBLICATIONS OF THE BUREAU OF MINES.

BULLETIN 1. The volatile matter of coal, by Horace C. Porter and F. K. Ovitiz. 1910. 56 pp. 1 pl.

Bulletin 2

DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

JOSEPH A. HOLMES, DIRECTOR

NORTH DAKOTA LIGNITE
AS A
FUEL FOR POWER-PLANT BOILERS

BY

D. T. RANDALL

AND

HENRY KREISINGER



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NORTH DAKOTA LIGNITE AS A FUEL FOR POWER-PLANT BOILERS.

By D. T. RANDALL and HENRY KREISINGER.

INTRODUCTION.

Importance of the work.—The tests described in the following report were made by the United States Geological Survey in cooperation with the United States Reclamation Service, which, in connection with its Williston project, had erected a large pumping plant at Williston, N. Dak., and had installed steam boilers with furnaces designed to burn a "brown lignite" that was mined on adjacent government land. The tests are deemed important because the lignite deposits of the Northwest are so extensive and the distance of the region from other coal fields is so great that a large portion of the United States, including parts of North Dakota, South Dakota, and Montana, may be greatly benefited by any improvements in the methods of utilizing this local fuel supply. The desirability of getting all available information regarding the methods of using similar lignite is obvious.

The lignite in this field is low in heating value, some of it containing nearly 45 per cent of its weight in moisture, and it is difficult to burn in the furnaces commonly used for the better grades of coal, but the tests have shown the possibility of designing suitable furnaces for burning it profitably. The results of tests of lignite on the hand-fired grates under a Heine boiler at the fuel-testing plant at St. Louis were not fully satisfactory in themselves, but showed the necessity of observing certain conditions for efficiently burning this fuel. They indicated that strong draft and a large combustion chamber are essential to insure the burning of the large quantity of gas that is driven off with the moisture from the fuel bed.

The results of the tests made at Williston, set forth in this report, show that this fuel, though generally considered unsatisfactory, may be used with fair economy under boilers that generate their full rated capacity. In fact, when the number of heat units available is con-

sidered the results compare very favorably with those of better grades of fuel.

Personnel.—The tests were made under the personal direction of Henry Kreisinger, who was assisted by F. E. Woodman, of the Geological Survey, and C. E. Draper, electrical assistant of the Reclamation Service. The furnace temperature measurements were made by J. K. Clement and the flue gases were sampled and analyzed by E. J. Hoffman, both of the Survey. The chemical work on coal and ashes was done at the chemical laboratory of the Geological Survey at Pittsburg, Pa., under the direction of F. M. Stanton. The computations were made at Pittsburg under the direction of Lauson Stone.

Acknowledgments.—Credit is due to A. A. Storrs and G. O. Sanford, the Reclamation Service engineers directly in charge of the Williston project, who had all changes made in the boiler-plant equipment that were necessary for the tests. J. M. Fine, engineer in charge of the power plant, and John G. Cunningham both rendered valuable assistance in connection with the operation of the boiler under test.

Related investigations.—The fuel researches of the Geological Survey, which had their beginning under an appropriation made by Congress on February 4, 1904, for testing fuel at the Louisiana Purchase Exposition, embraced not only the field study of mineral-fuel deposits, but also analyses and tests to determine the quality of the deposits, their adaptability to specific uses, and the methods by which they could be utilized to best advantage. In the course of these investigations the Survey collected and analyzed thousands of samples and made a great number of tests, such as washing, briquetting, coking, steaming, and producer-gas tests, to ascertain the fuel value of the peat, lignite, or coal under fixed conditions, its relative suitability for a given purpose, and the possibility of increasing its efficiency. Results of these tests are given in many of the bulletins listed at the end of this report.

The act of Congress of May 16, 1910, which established a Bureau of Mines, transferred to this bureau the work of analyzing and testing fuels that was being carried on by the Geological Survey. The act became effective July 1, 1910. Hence the Bureau of Mines is publishing a considerable number of reports, of which this is one, dealing with investigations made by the Geological Survey prior to that date.

Objects of the tests.—The primary object of the tests was to show that the lignite coals of North Dakota are suitable for making steam. The secondary objects were to determine the effect of the size and the weathering of coal on economy and to determine whether anything is gained by superheating the steam used in the Argand steam blowers.

FUEL, APPARATUS, AND METHODS.**THE FUEL TESTED.**

The fuel used on the tests came from the government mine located within about 500 yards of the power house. The mine is working the middle seam of the three known beds. The coal is a "brown lignite," containing about 40 per cent of moisture and 7 to 8 per cent of ash. It is therefore considered low-grade coal. In most of the tests the coal was burned soon after being mined. It was delivered in run-of-mine form to the power plant, where it was crushed either by a power crusher or by hand.

APPARATUS USED.**THE BOILER.**

The tests reported in this bulletin were made with one of six Stirling boilers installed in the power plant of the Williston irrigation project of the United States Reclamation Service. This boiler was located at the north end of the boiler room and was designated at the plant as No. 6. Boilers Nos. 5 and 6 were erected as a single battery.

Plate I shows two sections of the boiler and the setting. The boiler proper is of the standard Stirling water-tube type, consisting of three steam drums, one mud drum, and three nests of tubes. The baffles are inserted in the usual places.

THE FURNACE.

The furnace is of the semigas-producer type and has an external resemblance to the so-called Dutch oven. The most striking features in the construction of the furnace are the deep-set grate and the contraction of the space between the bridge wall and the end of the prolonged fire-brick arch. As shown in the plate, the grate is 21 inches below the lower edge of the firing door. The object of this construction is to enable the fireman to carry a thick fuel bed and still see its condition at the top. The side furnace door shown in the plate, used in removing clinkers, is only 7 inches above the grate.

The furnace is designed to work on the gas-producer principle. The solid fuel is gasified on the grate and the gas passes through the space under the arch into the combustion chamber, where most of the gaseous combustible burns. The necessary air for combustion is added through the openings A in the bridge wall. This air is preheated to 200° to 300° F. in coils P, and forced into the furnace under a pressure of 0.5 to 1 inch of water. Owing to the location and direction of these air openings, the air is blown in jets into the comparatively slow-moving body of combustible gas, thereby causing considerable stirring, so that the gases and the air form a fairly homogeneous mixture. A small quantity of air is also added through the

openings R in the roof of the furnace. This air consists only of leakage through the outside wall into the air space which completely envelopes the fire-brick lining of the Dutch-oven part of the furnace. Although there is some combustion above the fuel bed, the greater part of the gases burn below the contracted arch and back of the bridge wall, after air has been added through the openings A and R. With rates of combustion exceeding about 25 pounds of coal per square foot of grate the flames extend even into the space above the arch.

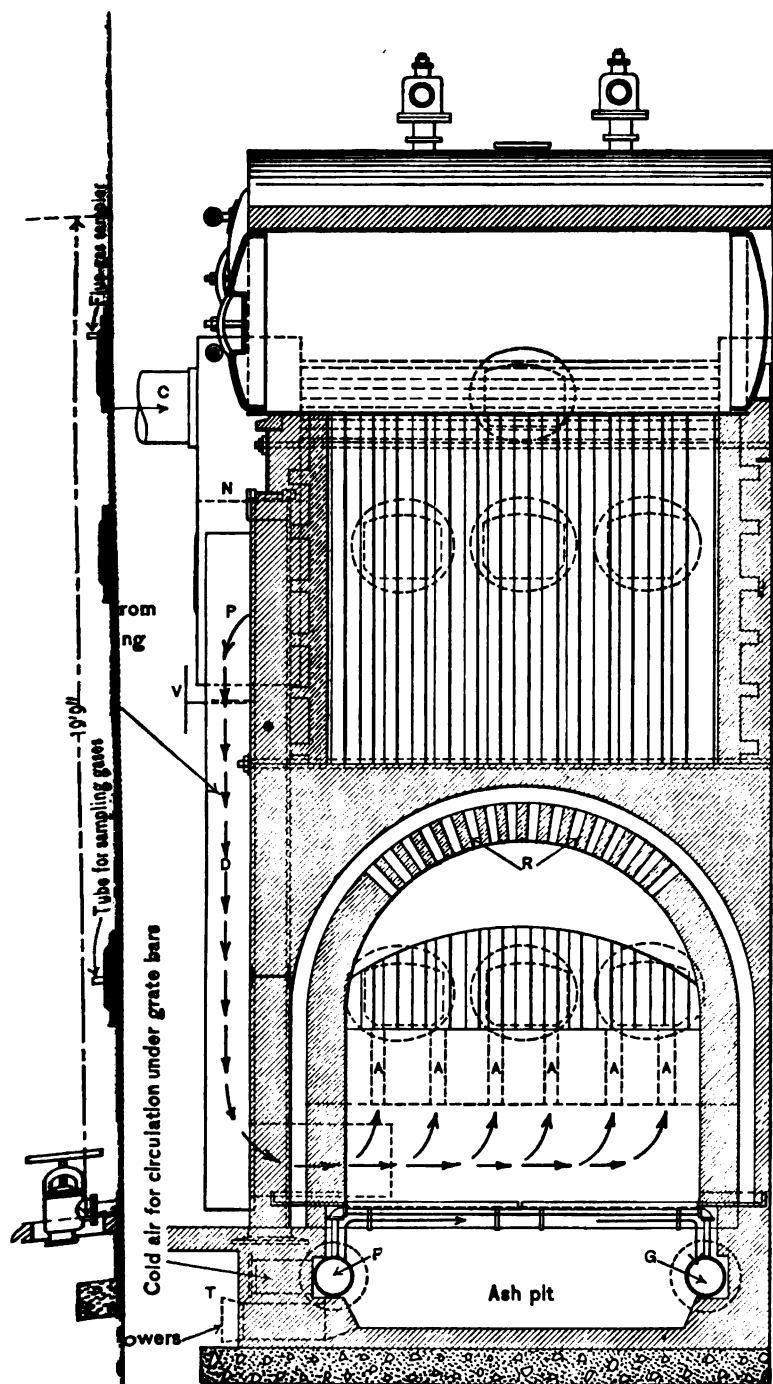
The air added through the bridge wall is forced by a fan blower through the duct C into the upper half, N, of the preheating coils. Through these coils the air passes to the other side of the boiler setting, where it is turned by means of a metal-sheet chamber into the lower half, P, of the preheating coils, through which it flows back to the same side of the boiler from which it started; it there enters the air duct D, which takes it into the upper tunnel in the bridge wall. A damper V placed in the upper part of the air duct D serves to regulate the amount of air to be put through the bridge-wall opening.

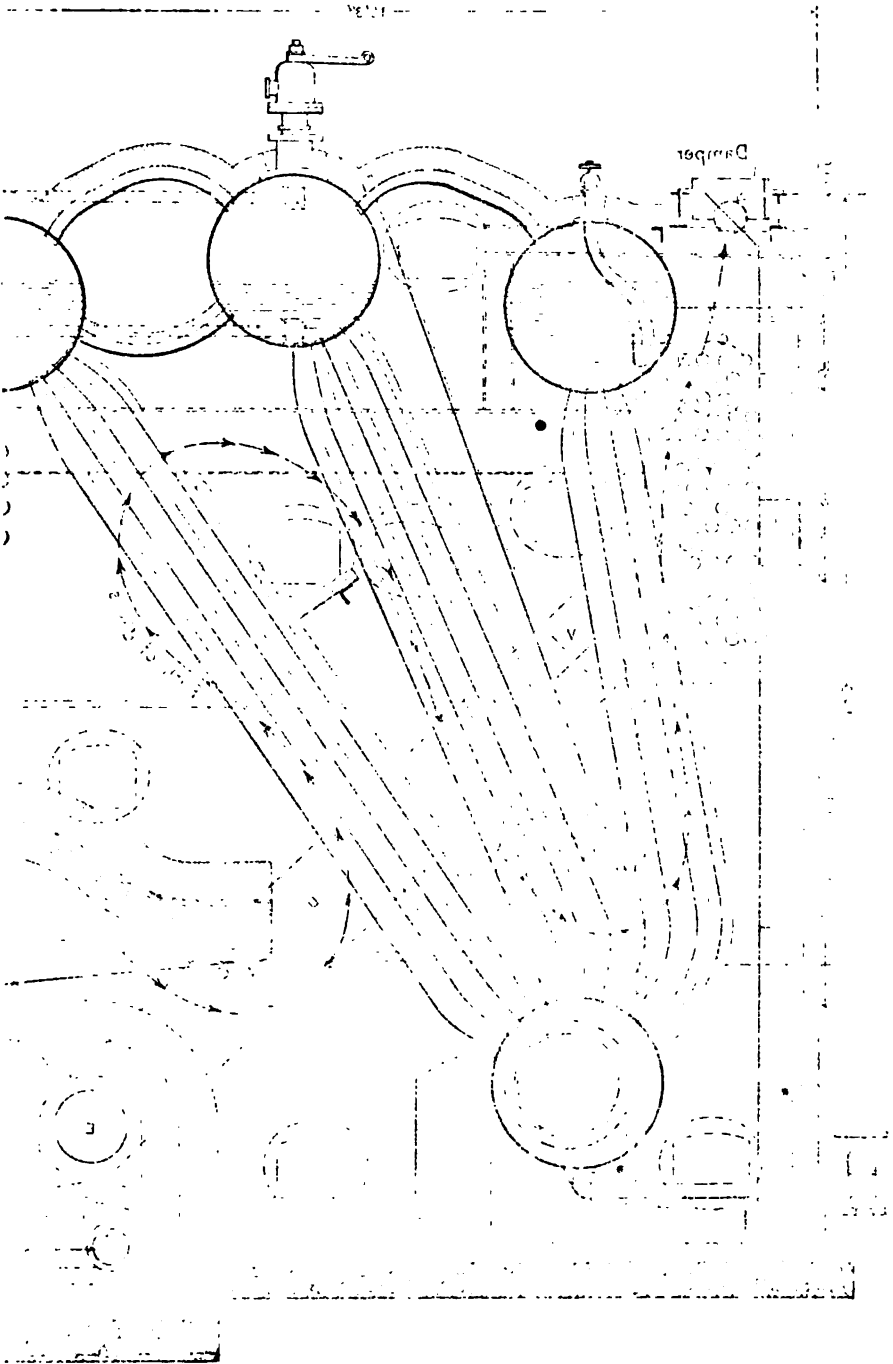
THE GRATE.

The furnace is equipped with a rocking grate consisting of two rows of grate bars. Each row is connected with two shaking levers in such a way that by shaking one lever alternate bars in the row are rocked. Figure 1 shows top and cross-sectional views of a grate bar. Each grate bar is supported by a 2-inch pipe, through which cold air circulates; the object is to keep the grate bars and the ash immediately above them cool, so that the ash will not fuse and adhere to the bars. Cold air from the fan blower is brought through a special air duct into the large pipe F, from which it flows through each grate-supporting pipe into the pipe G, as indicated by the arrows in Plate I. Pipe G takes the air into the lower tunnel H in the bridge wall, through which the air flows into the pit T, and is thence blown by the Argand blowers into the ash pit. The pressure in the ash pit varies from 1 to 2 inches of water; the pressure whereby the cold air circulates through the supporting bars is only about 0.5 to 1 inch of water, so that the circulating air can not be discharged from the supporting bars (pipes) directly into the ash pit.

STACK AND BLOWERS.

Boiler No. 6 has a steel stack in common with boiler No. 5. Each boiler has a separate damper placed in the brick gas passage immediately below the hood of the stack. The path of gases through the boiler is indicated in Plate I by the long curved arrow. The boiler is operated with "balanced draft"—that is, the ash pit is kept under pressure higher than the atmospheric and the uptake under a pres-





sure a little below atmospheric; the furnace is, therefore, just about at atmospheric pressure. This condition permits the opening of the fire doors when firing without allowing outside air to enter.

A coil used for superheating the steam used by the Argand blowers is inserted in the space above the arch, directly under the first steam drum.

Ordinarily, the steam used in the two Argand blowers is taken from the same boiler the blowers are operating. However, on these

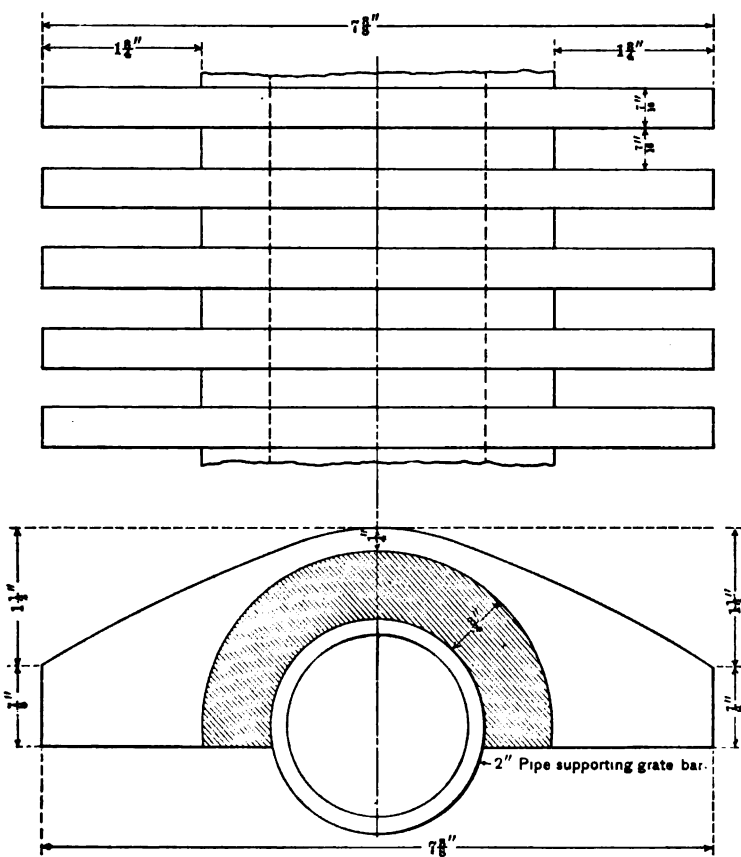


FIGURE 1.—Top view and vertical cross section of grate bar, showing construction.

tests, in order to determine the amount of steam used by the blowers, the steam was supplied by an independently operated small locomotive boiler of 20-horsepower capacity. The connections between the locomotive boiler and the blowers were such that the steam could either be made to pass first through the superheating coils inserted in the main boiler setting or could be used directly in its saturated state.

DIMENSIONS.

Table 1 gives the principal dimensions of the furnace, the boiler, and the grate.

TABLE 1.—Principal dimensions of furnace, boiler, and grate.

Furnace:			Firing door:		
Width in front.....	feet..	6.6	Lower edge above grate.....	inches..	21
Width back of bridge wall.....	do....	7.5	Height.....	do....	14.75
Length.....	do....	8	Width.....	do....	19.75
Height.....	do....	7	Chimney:		
Roof of furnace:			Height above grate.....	feet..	155
Length (straight portion).....	do....	8.5	Diameter.....	inches..	54
Length at sides.....	do....	15.8	Boiler:		
Length in middle.....	do....	15.3	Builders' rating.....	horsepower..	258
Height in rear above bridge wall.....	inches..	23	Water-heating surface.....	square feet..	2,587
Number of openings ^a		16	Diameter of steam drums.....	inches..	42
Size of openings.....	inches..	2 by 4.5	Length of steam drums.....	feet..	10.33
Width of partitions.....	inches..	2	Diameter of mud drums.....	inches..	42
Bridge wall:			Length of mud drums.....	feet..	8.83
Width at base.....	feet..	5	Number of tubes.....		209
Width on top.....	do....	3	Diameter of tubes.....	inches..	3.25
Height.....	inches..	40	Grate:		
Number of openings ^b		6	Width.....	feet..	6.6
Size of openings.....	inches..	5 by 2.5	Length.....	do....	8
Side door:			Width of grate bars.....	inches..	7.4
Lower edge above grate.....	inches..	7.5	Width of rib.....	do....	.44
Height.....	do....	17	Width of air space.....	do....	.44
Width.....	do....	19	Depth of air space.....	do....	1.75

FLUE-GAS SAMPLER.

Figure 2 shows a flue-gas sampling device placed in the uptake about 18 inches below the damper. Plate I shows the location of the sampler with respect to the boiler in a vertical section. The gas sampler is a standard $\frac{3}{4}$ -inch iron pipe closed at both ends with caps and having two rows of small ($\frac{3}{32}$ -inch) holes 4 inches apart drilled in a staggered way on two sides of the pipe. This $\frac{3}{4}$ -inch pipe connects in the middle to a $\frac{1}{4}$ -inch pipe which extends out of the setting and leads down to within 3 feet of the floor, where it is connected to a steam ejector. Figure 3 shows the steam ejector and connections. The sample of gas for chemical analysis is taken through the pet cock, as shown in the figure. The object of the steam ejector is to induce a continual stream of gas to flow from the uptake through the sampling apparatus. The construction of the $\frac{3}{4}$ -inch pipe placed in the uptake permits of drawing the gas from a large area, so that the sample drawn represents fairly well the average composition of all the gases passing through the uptake.

METHODS OF CONDUCTING TESTS.

Starting and closing.—A modification of the "alternate method" was used in starting and closing tests. The standard method was

^a Openings in roof of furnace are 6.5 feet from front of furnace.

^b Openings in bridge wall are 9 inches from front of bridge wall.

altogether out of question. The fuel bed is considered to be in its best condition when it is 18 to 22 inches thick; when a new fire is started with wood it takes several hours to get the bed built up and the fire in a good running condition.

Generally, a fire was started with wood about 4 o'clock in the morning and the bed was gradually built up until about 8 or 9 o'clock, when it was 16 to 18 inches thick. At this time the furnace was well heated and in condition for a test to be started. The grates were then shaken and all ash removed from the ash pit. At the time of starting and at the time of closing each test the height of the fuel bed was estimated as accurately as possible by sighting over the bottom of the opening of the firing door to see how far the upper surface of the fuel bed was below this line. This estimate of the fuel bed was made independently by at least two observers; if the observations did not agree, the average of the estimates was taken for record. Usually C. E. Draper and Henry Kreisinger did this estimating.

At the end of test 2 the fuel bed was burned down to the same height as at the start. It was thought that only an inappreciable amount of clinker would be found in the fuel bed, but, after closing the test and burning the fuel bed down still farther, a quantity of clinkers amounting to several hundred pounds was found on the grate. It was evident that to neglect these clinkers would introduce considerable error in the results of the tests; therefore, on all subsequent tests corrections for the clinkers were made by closing the test with the fuel bed 3 to 4 inches higher than it was when the test was started. This method of making allowance for clinker was necessary because the clinkers could not be pulled out during the test on account of the grate being so far below the firing door. Care was taken also to have the height of water in the boiler and the steam pressure as nearly as possible the same in closing as at the start.

Ash and refuse.—Immediately before starting each test the grate was shaken and the ash pit was cleaned out. A few minutes before closing the test the grate was again shaken. After closing the test all the ash was removed from the ash pit, weighed, and charged to the test. The fuel bed was then burned down to about 12 to 14 inches and the clinkers were pulled out, partly through the side door and partly through the firing door. This process of pulling out the clinkers was very difficult, as the firing doors were high above the grate and there was not enough room on the side of the boiler for handling the fire tools through the side door. The removal of clinkers took from one to two hours. Only the larger pieces of clinker could be pulled out; smaller pieces fell through the grate into the ash pit with free ash and hot coal and could not be separately recovered. Frequently during the process of pulling clinkers large pieces were so broken as to be much reduced in size before removal. Coal,

ash, and small pieces of clinker falling into the ash pit were not weighed. The method of obtaining the weight of ashes and clinkers was very crude, but it was the best that could be done under the circumstances. Taking all factors into consideration, it is estimated that only about one-half to two-thirds of the total weight of clinkers originally present on the grate were removed, weighed, and charged to the test.

Weighing and sampling fuel.—Coal was weighed in a wooden box which was placed on a platform scale. The box was 4 feet long by 3 feet wide by 14 inches deep, and held about 500 pounds of coal. One side of the box was removable for convenience in shoveling. The time of firing and the weight of coal at each charge were regularly recorded.

Each time the box was filled a sample of coal amounting to about a shovelfull was collected by taking a little at a time.

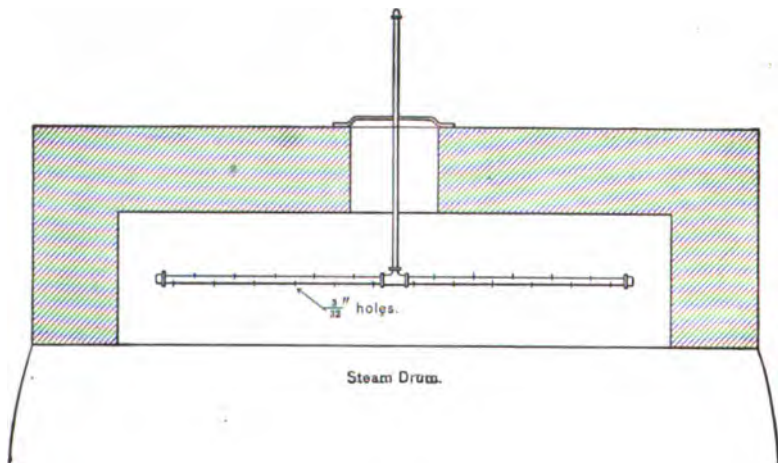


FIGURE 2.—Section through uptake, showing location of flue-gas sampler.

Feed water.—Water was measured in a specially constructed cylindrical tank, which held 1,293 pounds of water, at 60° F. The tank was about 2½ feet in diameter by 5 feet high and had a funnel-shaped bottom which permitted quick emptying. The top of the tank was contracted by means of an inverted funnel to a neck 5 inches in diameter. This narrow neck at the top of the tank reduced the error which might have been introduced by any possible difference in the water level when filling the tank.^a The measuring tank was placed on a wooden platform about 7 feet high and emptied into a plain cylindrical suction tank placed on the boiler-room floor. This suction tank was of larger capacity than the measuring tank. Water was fed from the suction tank into the boiler by an injector (1½ inch), the overflow from which was run back into the suction tank. The

^a For details of construction of a similar tank see Professional Paper U. S. Geological Survey No. 48, 1906, p. 309.

capacity of the suction tank was determined for every inch of the height of water. The time of each emptying of the measuring tank was recorded.

During the tests all pipe lines except the injector were disconnected from the boiler. In disconnecting the blow-off pipe from the main line its end was left exposed so that it was at all times visible. These arrangements made it impossible for any water to get into the boiler, except that which was measured and fed into the boiler with the injector, or for any water to get out of the boiler except through the steam main in the form of steam. During each test the rate of combustion was kept as constant as possible, the drafts being changed if necessary. A scale was attached to the gage glass of the test boiler and the height of the water was read at regular twenty-minute intervals.

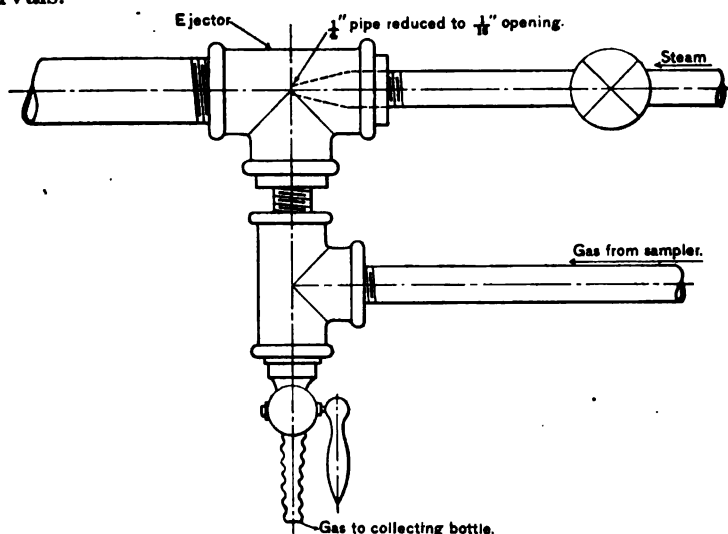


FIGURE 3.—Steam ejector, showing its location at the end of the pipe leading from gas sampler in stack.

Flue gas sampling.—Samples of flue gas were collected and analyzed for CO_2 , O_2 , and CO with an Orsat apparatus. Each regular sample was collected through the sampling device (illustrated in figures 2 and 3) during a period of about thirty minutes, so that the analysis of the sample represented the average chemical composition of the gases during that time.

Occasionally a sample of gas was taken with a $\frac{1}{4}$ -inch iron pipe through a hole in the lowest middle door in the rear of the boiler setting. The end of this sampling pipe was placed halfway between the mud drum and the baffle between the second and third nests of boiler tubes.

Temperature measurements.—Except as otherwise stated, temperatures were taken for record at regular twenty-minute intervals. At

tempts were made at first to measure flue-gas temperatures in the uptake at the place where the regular gas sample was taken, but owing to the close proximity of the air preheating coils and the steam drum, the thermometer gave very low readings. The upper half of the preheating coils received the cold air from the fan. These coils were therefore much colder than the flue gases going up the chimney. The thermometer being close to these cold coils radiated heat to them and consequently showed much lower temperatures than the flue gases passing by it. So it was decided to put a thermometer about 4 feet above the damper in the hood of the stack. The thermometer read too low also in this position, because the hood was much colder than the gases and of course the thermometer radiated heat to it. Moreover, the two boilers in the battery had a stack and hood in common so that the colder gases from boiler No. 5, under which only a very low fire was generally kept, mixed somewhat with the gases from the test boiler. In spite of these cooling effects the thermometer in the hood of the stack read 100° to 200° F. higher than the thermometer at the point below the damper. It is an open question how much too low the indicated temperatures in the hood were, but there can be no doubt that the gases leaving the boiler setting were hotter than is shown by the record of thermometer readings. The thermometer used for this purpose was a large mercury thermometer made especially for measuring flue-gas temperature.

The temperature of water was determined at each time of emptying the tank by means of a thermometer immersed therein.

The temperature of the preheated air which was fed into the furnace through the bridge wall was measured with a mercury thermometer inserted in the air duct D (Pl. I) about 5 feet from its lower end.

A thermometer cup was placed in the steam pipe leading from the superheater to the Argand blowers and a mercury thermometer was used for measuring the temperature of the superheat of the steam used.

Furnace temperatures were measured with a Wanner optical pyrometer. The measurements were made through a hole (W, Pl. I) bored in the side wall just back of and a little above the bridge wall.

Steam.—Steam pressures were recorded at regular intervals from a standard gage which was calibrated and could be read within 1 pound. A throttling calorimeter for moisture determinations was attached in the approved way to the steam pipe leading from the boiler to the steam header. The steam used by the Argand blowers was furnished by a separately operated locomotive boiler. The water fed into this boiler was weighed and charged to the blowers.

Measuring drafts.—Gas pressures (drafts) were measured on all tests, in the ash pit, over the fuel bed, in the uptake, and in the preheated-air duct by means of Ellison inclined tube manometers, and readings were taken at regular twenty-minute intervals.

ACCURACY OF DATA.

Perhaps the greatest error introduced in the data of the tests was in estimating the height of the fuel bed and in allowing for the clinkers in the fuel bed at the end of the tests. It is not probable, however, that the error in any test was over 2 inches in the net thickness of fuel bed.

The grate area is approximately 54 square feet, and 1 cubic foot of lignite weighs about 50 pounds, so that a difference of 2 inches in the height of the fuel bed would be equal to 9 cubic feet, or $9 \times 50 = 450$ pounds of coal.

About 28,000 pounds of coal were burned during each test. An error of 2 inches in estimating the height of the fuel bed would therefore be equal to about 1.6 per cent of the total coal used. The estimated height of the fuel bed might be either 2 inches too low or 2 inches too high, so that there is a possibility of the results of tests given in this report being 1.6 per cent too high or too low. Thus, for instance, if the boiler efficiency item 72* is given as 60 per cent, the margin of doubt is $0.60 \times 1.6 = 0.96$, or about 1 per cent either way. In other words, the boiler efficiency may be anywhere from $60 - 1 = 59$ to $60 + 1 = 61$ per cent.

The flue-gas temperatures as recorded, on account of conditions already stated, are very likely too low; but such error affects only the heat balance and not the economic results. A flue-gas temperature that is too low lowers items 2, 3, and 4 of the heat balance and raises item 6 by the same amount. It does not affect the evaporation or the boiler efficiency.

Thus, for example, in test 4 the stated flue-gas temperature is 436° F. It is possible that the actual temperature was 50° F. higher, or 486° F. If so, the corresponding four items of the heat balance for the two temperatures are as follows:

TABLE 2.—Values of items of heat balance corresponding to different degrees of flue-gas temperature.

Flue-gas temperatures ($^{\circ}$ F.).	Items in heat balance (expressed in percentage of heat in 1 pound of combustible).			
	2	3	4	6
As stated, 436.....	9.47	4.60	12.62	11.78
Hypothetical, 486.....	9.80	4.83	15.10	8.74

The second line is perhaps closer to the true values than the first one.

DATA AND RESULTS OF TESTS.

TABULATED ITEMS.

Fifteen tests were made. In tests 8, 9, 10, and 12 saturated steam was used in the Argand blowers; for all other tests superheated steam was used. In tests 5, 6, and 7 coal crushed by hand was used; coal for all other tests was crushed by crusher. Coal used in test 13 was exposed to wind and sun for forty-eight hours. Coal used in test 14 was exposed twenty-four hours. The averages of the principal items of observed data and the calculated results are given in Table 3. The computations embodied in the table were made according to the method explained in Bulletin 325.^a This method is in practical accord with the recommendations of the American Society of Mechanical Engineers. The figures in parentheses at the heads of columns in the table are the item numbers of the society's code for steaming tests.

The economic results of the tests are given in columns 51-53, 56, and 66-69 of Table 3.

The three different efficiencies (columns 67-69) express three different ratios. Item 72* (column 67), efficiency of boiler, is the ratio of the heat absorbed by boiler to the heat of the combustible ascending from the grate. It is the efficiency of the furnace and boiler, but not of the grate. Combustible falling through the grate is not consumed and is excluded from this efficiency. Item 72*, expressed in per cent, corresponds to column 78 under "Heat balance," where the value in British thermal units is stated.

Item 73 (column 68) is the combined efficiency of the boiler, furnace, and grate. It is the ratio of the heat absorbed by the boiler to the heat in all the coal fired on the grate. Item 72* multiplied by the efficiency of the grate gives item 73.

The column 69 of Table 3 shows the "over-all efficiency of the steam generating apparatus," including the ash-pit blowers. This efficiency is the ratio of the heat absorbed by the boiler minus the heat of steam used in ash pit divided by the heat of the coal fired on the grate, and is equal to item 73 minus the heat of steam used in ash-pit blower, the heat being expressed in percentage of the heat in dry coal.

^a Breckenridge, L. P., A study of four hundred steaming tests: Bull. U. S. Geol. Survey No. 325, 1907, pp. 151-152.

TABLE 3.—Summary of observed data and calculated items of 15 steaming tests made with North Dakota lignite, October 8-29, 1908.^a

Test No.	Time of test.		Condition of fuel.	Average pressures.			Draft (inches of wafer).			
	Date, October, 1908.	Duration (hours).		Barometer (inches of mercury).	Steam, above atmosphere (lbs per sq. in.).		Below atmosphere.		Above atmosphere.	
					At gage.	To ash pit.	Stack.	Furnace.	Ash pit.	To bridge wall.
1	2 (1)	3 (2)	4	5 (11)	6 (11.1)	7	8 (12)	9 (13)	10	11
1.....	8	3.18	Fresh ^b	28	141.4	0.61	0.12	0.66
2.....	10	13.77	do. ^b	28.18	134.6	67.9	.15	.75	.90	0.17
3.....	11	10.47	do. ^b	28.33	131.9	77.9	.23	.10	1.40	.16
4.....	12	14.75	do. ^b	28.16	133.1	49.7	.13	.11	1.42	.17
5.....	14	14.25	do. ^c	27.54	133.9	51.2	.18	.14	1.55	.13
6.....	16	11.13	do. ^c	27.69	130.7	68.9	.24	.15	1.78	.32
7.....	17	12.70	do. ^c	28.19	134.8	63.6	.11	.09	1.32	.42
8.....	19	10.00	do. ^b	27.72	135.1	58.3	.24	.13	.93	.27
9.....	20	12.95	do. ^b	27.12	131.0	64.5	.21	.16	1.67	.33
10.....	21	14.67	do. ^b	27.75	131.6	60.6	.16	.12	1.37	.38
11.....	22	10.03	do. ^b	28.30	135.2	105.9	.41	.21	1.74	.36
12.....	23	11.00	do. ^b	28.21	135.0	104.8	.32	.18	1.83	.73
13.....	26	13.42	Weathered ^d	28.13	131.4	92.0	.30	.20	1.78	.46
14.....	27	12.73	do. ^e	27.96	131.3	86.6	.27	.17	1.95	.54
15.....	29	12.12	Fresh ^b	28.31	133.5	82.7	.31	.17	2.35	.43
15 A.....	29	8.13	do. ^b	28.31	133.238	.20	2.27

Test No.	Average temperature (*F.) of—								Fuel (total weights in pounds).		
	Atmosphere.		Pre-heated air to bridge wall.	Steam to ash-pit blowers.	Feed water in tank	Calorimeter.	Flue gas.	Furnace.	As fired.	Dry.	Ash and refuse.
	Outside.	Fire-room.									
1	12	13 (16)	14	15	16 (18)	17	18 (21)	19	20 (25)	21 (27)	22 (28)
1.....	71	85	446	55	299	442	1,791	6,350	258
2.....	44	75	271	557	54	291	373	2,181	30,729	17,897	1,540
3.....	59	83	279	533	53	292	457	2,171	29,000	16,527	1,558
4.....	66	84	240	455	54	291	436	2,014	27,998	15,606	2,198
5.....	70	88	251	532	56	287	474	2,145	28,000	16,895	1,170
6.....	49	72	246	488	60	276	519	2,160	28,340	16,536	1,103
7.....	40	66	217	463	57	291	494	2,122	27,101	15,705	781
8.....	45	68	241	327	51	291	521	2,080	28,000	15,725	1,986
9.....	45	67	218	332	55	285	472	2,109	28,500	15,837	1,665
10.....	37	68	207	332	43	285	462	2,148	28,000	15,764	1,586
11.....	39	69	242	39	286	570	2,073	28,000	15,994	1,997
12.....	46	65	210	336	48	290	549	2,166	30,000	17,271	1,084
13.....	50	65	197	483	54	276	500	2,115	30,000	17,469	1,079
14.....	51	75	210	507	48	288	528	2,161	27,997	16,129	1,305
15.....	33	66	223	549	41	280	526	2,120	31,672	18,167	2,391
15A.....	35	67	38	282	555	2,156	21,659	12,424	1,635

^a Code numbers (in parentheses at the top of certain columns) refer to corresponding items explained in Bull. U. S. Geol. Survey No. 325, pp. 151-153. See also Prof. Paper U. S. Geol. Survey No. 48, pt. 2.

^b Crushed by crusher; 20 to 30 per cent slack.

^c Crushed by hand.

^d Coal exposed to wind and sun for forty-eight hours.

^e Coal exposed to wind and sun for twenty-four hours.

/ Condition of steam used in ash-pit blowers: Saturated in tests 8, 9, 10, and 12; superheated in all other tests.

TABLE 3.—*Summary of observed data and calculated items of 15 steaming tests made with North Dakota lignite, October 8-29, 1908—Continued.*

Test No.	Total combustibles consumed (lbs.).	Fired per hour (lbs.).			Refuse in dry fuel (per cent.).	Proximate analysis (per cent.).						
		Dry fuel.		Combustible.		Fixed carbon.		Volatile matter.		Moisture in fuel as fired.	Ash in fuel.	
		For grate.	Per sq. ft. of grate.			In moist coal.	In combustible.	In moist coal.	In combustible.		As fired.	Dry.
1	23 (30*)	24 (46)	25 (48)	26 (47*)	27 (31)	28 (32)	29 (32a)	30 (33)	31 (33a)	32	33	34
1.....												
2.....	15,423	1,300	23.99	1,120	8.60	28.01	54.99	22.93	45.01	41.76	7.30	12.53
3.....	14,068	1,579	29.13	1,343	9.43	26.01	52.30	23.72	47.70	43.01	7.26	12.75
4.....	12,909	1,058	19.62	875	14.08	24.00	49.48	24.51	50.52	44.26	7.23	12.97
5.....	14,860	1,186	21.88	1,043	6.93	27.56	51.02	26.46	48.98	39.66	6.32	10.47
6.....	13,909	1,486	27.42	1,250	6.67	25.75	50.96	24.78	49.04	41.65	7.82	13.40
7.....	13,428	1,237	22.82	1,057	4.97	25.82	51.72	24.10	48.28	42.05	8.03	13.86
8.....	13,237	1,573	29.02	1,324	12.63	24.86	50.79	24.08	49.21	43.84	7.22	12.86
9.....	13,648	1,223	22.56	1,054	10.51	24.65	50.45	24.22	49.55	44.43	6.70	12.06
10.....	13,678	1,075	19.83	932	10.06	25.28	50.98	24.31	49.02	43.70	6.71	11.92
11.....	13,038	1,595	29.43	1,300	12.49	24.96	50.76	24.22	49.24	42.88	7.94	13.90
12.....	15,164	1,570	28.97	1,379	6.27	26.17	51.34	24.80	48.66	42.43	6.60	11.46
13.....	15,164	1,302	24.02	1,130	7.49	28.22	54.96	23.13	45.04	41.77	6.88	11.82
14.....	14,281	1,267	23.38	1,122	6.67	27.94	54.56	23.27	45.44	42.39	6.40	11.11
15.....	15,584	1,499	27.66	1,286	13.16	27.41	53.93	23.42	46.07	42.04	6.53	11.38
15A.....	10,656	1,528	28.19	1,311	13.16	27.41	53.93	23.42	46.07	42.64	6.53	11.38

Test No.	Sulphur, separately determined (per cent.).			Ultimate analysis (per cent.).							
	In moist coal.	In dry coal.	(b)	Carbon.		Hydrogen.		Oxygen.		Nitrogen.	
				Dry coal.	(b)	Dry coal.	(b)	Dry coal.	(b)	Dry coal.	(b)
1	35	36 (41)	37 (41a)	38 (37)	39 (37a)	40 (38)	41 (38a)	42 (39)	43 (39a)	44 (40)	45 (40a)
1.....											
2.....	0.42	0.72	0.82	60.15	68.77	4.35	4.97	21.18	24.22	1.07	1.22
3.....	.60	1.07	1.22	59.76	68.49	4.32	4.95	21.04	24.12	1.06	1.22
4.....	.79	1.42	1.63	59.36	68.20	4.29	4.93	20.90	24.02	1.06	1.22
5.....	.63	1.04	1.16	61.36	68.53	4.42	4.94	21.62	24.15	1.09	1.22
6.....	1.29	1.39	1.61	59.09	68.23	4.27	4.93	20.80	24.02	1.05	1.21
7.....	1.71	1.86	2.16	58.45	67.85	4.23	4.91	20.56	23.87	1.04	1.21
8.....	1.00	1.09	1.25	59.67	68.48	4.31	4.95	21.00	24.09	1.07	1.23
9.....	1.20	1.30	1.48	60.09	68.33	4.34	4.94	21.13	24.02	1.08	1.23
10.....	1.01	1.10	1.25	60.32	68.48	4.35	4.94	21.23	24.10	1.08	1.23
11.....	1.53	1.68	1.85	58.54	67.99	4.24	4.92	20.59	23.92	1.05	1.22
12.....	1.22	1.32	1.49	60.48	68.31	4.36	4.92	21.30	24.06	1.08	1.22
13.....	.96	1.05	1.19	60.41	68.51	4.36	4.94	21.28	24.14	1.08	1.22
14.....	.84	.92	1.04	61.00	68.63	4.41	4.96	21.47	24.14	1.09	1.23
15.....	1.18	1.31	1.48	60.55	68.32	4.38	4.94	21.30	24.04	1.08	1.22
15A.....	1.18	1.31	1.48	60.55	68.32	4.38	4.94	21.30	24.04	1.08	1.22

a For explanation of method of computing this item see p. 27.

b Accompanying 100 per cent of "combustible" (coal, moisture and ash free).

TABLE 3.—Summary of observed data and calculated items of 15 steaming tests made with North Dakota lignite, October 8-29, 1908—Continued.

Test No.	Water fed to boiler (pounds).									Evaporation.		
	Total.	Equivalent evaporated from and at 212° F.							Actually evaporated. ^a		(d)	Factor of.
		Total.	Per hour.	(b)	Into dry steam.	Per pound of fuel—			Total.	Per hour.		
						As fired.	Dry.	(c)				
1	46 (57)	47	48 (63)	49	50 (61)	51 (69)	52 (70)	53 (71)*	54	55	56 (68)	57 (60)
1.....	18,410	22,285	6,990	2.70	22,227	3.50	18,362	5,774	2.90	1.2105
2.....	94,187	114,004	8,234	3.18	113,377	3.69	6.33	7.35	93,669	6,802	3.07	1.2104
3.....	82,450	99,847	9,494	3.46	99,407	3.43	6.01	7.07	82,087	7,840	2.84	1.2110
4.....	78,050	94,456	6,372	2.46	93,964	3.36	6.02	7.34	77,660	5,265	2.79	1.2102
5.....	88,404	106,810	7,445	2.88	106,094	3.79	6.28	7.14	87,812	6,262	3.16	1.2062
6.....	78,520	94,507	8,397	3.25	93,457	3.30	5.85	6.72	77,648	6,976	2.77	1.2036
7.....	82,043	99,059	7,757	3.00	98,514	3.63	6.27	7.34	81,592	6,425	3.03	1.2074
8.....	73,179	88,810	8,836	3.42	88,357	3.16	5.62	6.68	72,806	7,281	2.61	1.2136
9.....	82,137	99,287	7,614	2.94	98,602	3.46	6.23	7.22	81,570	6,299	2.88	1.2088
10.....	87,126	106,407	7,200	2.78	105,619	3.77	6.70	7.72	86,451	5,895	3.11	1.2213
11.....	80,162	98,287	9,725	3.76	97,540	3.49	6.10	7.48	79,553	7,931	2.86	1.2261
12.....	88,875	106,134	9,773	3.77	107,508	3.58	6.22	7.09	88,360	8,033	2.96	1.2167
13.....	92,103	111,435	8,210	3.17	110,176	3.67	6.31	7.27	91,062	6,786	3.07	1.2099
14.....	87,195	106,038	8,281	3.20	105,413	3.77	6.54	7.38	86,681	6,809	3.11	1.2161
15.....	79,135	96,845	7,911	3.06	95,887	3.03	5.28	6.15	78,352	6,465	2.50	1.2238
15A.....	59,557	73,065	8,908	3.44	72,422	3.34	5.83	6.80	59,033	7,261	2.75	1.2268

Test No.	Composition of refuse (per cent).			Heat value per pound (B. t. u.).		Steam (per cent).		Horsepower developed.		Efficiency (per cent). ^e		
	Car- bon.	Earthy matter.	Clink- er.	Dry fuel.	Combust- ible.	Mois- ture in.	Qual- ity of.	In boiler.	Per cent of rated.	Of boiler and furnace.		Over all.
										Alone.	Includ- ing grate.	
1	58 (44)	59	60 (29)	61 (50)	62 (51)	63 (54)	64 (56)	65 (65)	66	67 (72)*	68 (73)	69
1						0.35	99.74	202.6	81.0			
2	15.10	84.90	28.3	10,121	11,570	.75	99.45	238.7	95.5	61.35	60.39	
3	22.85	77.15	47.0	10,098	11,573	.60	99.56	275.2	110.1	58.99	57.47	
4	30.60	69.40	39.4	10,075	11,576	.68	99.50	184.7	73.9	61.23	57.70	52.50
5	22.75	77.25	20.6	10,397	11,612	.94	99.33	215.8	86.3	59.38	58.33	53.59
6	37.24	62.76	43.1	10,026	11,578	1.49	98.89	243.4	97.4	56.05	54.42	49.98
7	12.98	87.02	60.3	9,936	11,534	.74	99.45	224.8	90.0	61.46	60.94	55.52
8	23.45	76.55	47.4	10,116	11,610	.69	99.49	256.1	102.4	55.56	53.65	49.81
9	16.79	83.21	46.5	10,190	11,587	.93	99.31	220.7	88.3	60.17	59.04	53.70
10	13.04	86.96	49.7	10,222	11,605	1.01	99.26	208.7	83.5	64.24	63.30	57.88
11	36.64	63.36	34.9	99,949	11,554	1.05	99.24	281.9	112.8	62.52	59.21	54.46
12	11.68	88.32	49.8	10,258	11,585	.79	99.42	283.3	113.3	59.10	58.86	54.04
13	18.39	81.61	55.8	10,238	11,610	1.53	98.87	238.0	95.2	60.47	59.52	54.36
14	5.25	94.75	63.5	10,330	11,621	.80	99.41	240.0	96.0	61.33	61.14	55.82
15	21.56	78.44	38.8	10,271	11,590	1.36	99.01	229.3	91.7	51.24	49.64	
15A	21.56	78.44	38.8	10,271	11,590	1.21	99.12	258.2	103.3	56.66	54.81	

^a Corrected for quality of steam.^b Per hour per square foot of water-heating surface.^c Moisture and ash free.^d Apparent, per pound of coal as fired.^e Figured from chemical analyses of ash and coal.

TABLE 3.—Summary of observed data and calculated items of 15 steaming tests made with North Dakota lignite, October 8-29, 1908—Continued.

Test No.	Average thickness of fuel bed (inches).	Average intervals between firings (minutes).	Dry chimney gases.					Heat value of 1 pound of combustible (B. t. u.).	Heat balance. ^b		
			Per pound of combustible (pounds).	Analysis (per cent) ^a					Absorbed (1) by boiler (B. t. u.). ^c	Heat lost in dry flue gases (4).	
				CO ₂ .	O ₂ .	CO.	N ₂ .			B. t. u.	Pr. ct.
1	70 (81)	71 (82)	72	73 (84)	74 (85)	75 (86)	76	77	78	79	80
1.....	9-10	5.6		10.43	7.34	1.40	80.74				
2.....	10	4.6	16.94	10.04	9.04	.10	80.82	11,570	7,068	1,212	10.47
3.....	22	3.3	15.19	11.11	7.34	.19	81.36	11,573	6,827	1,363	11.78
4.....	18-20	5.0	17.29	9.79	9.27	.05	80.89	11,576	7,068	1,461	12.62
5.....	17	4.9	15.04	10.83	7.99	.58	80.80	11,612	6,895	1,392	11.99
6.....	22	3.6	14.29	11.18	7.42	.79	80.61	11,578	6,490	1,533	13.24
7.....	22-24	4.2	13.54	12.40	6.37	.23	81.00	11,534	7,068	1,390	12.05
8.....	18-20	3.5	13.88	12.23	6.45	.19	81.13	11,610	6,451	1,509	13.00
9.....	22	4.2	14.48	11.49	7.60	.36	80.55	11,587	6,972	1,407	12.14
10.....	20	5.0	15.09	11.06	8.20	.32	80.42	11,005	7,455	1,427	12.30
11.....	20	3.1	15.30	10.92	8.26	.23	80.59	11,554	7,223	1,837	15.90
12.....	18-20	3.2	13.58	12.14	6.46	.52	80.88	11,585	6,847	1,577	13.61
13.....	22	3.9	15.82	10.40	8.74	.43	80.43	11,610	7,021	1,652	14.23
14.....	22	4.1	14.84	11.50	7.83	.12	80.55	11,621	7,127	1,613	13.88
15.....	18-20	4.8	13.75	11.44	7.03	1.03	80.50	11,580	5,939	1,518	13.10
15A.....	18-20	4.8	14.15	11.54	7.12	.58	80.76	11,580	6,567	1,657	14.30

Test No.	Heat balance.								Boiler supplying steam to ash pit.					
	Loss due to moisture.				Loss due to incomplete combustion of carbon (5).		Loss in escaping hydrocarbons, radiation, and unaccounted for (6).		Time used (hours).	Used for Argand blowers.				
	In the fuel (2).		Formed by the burning of hydrogen (3).							Steam (pounds).		Heat in form of steam (per cent).	Evaporation (per cent).	
	B. t. u.	Pr. ct.	B. t. u.	Pr. ct.	B. t. u.	Pr. ct.	B. t. u.	Pr. ct.		Total.	Per hour.			
1	81	82	83	84	85	86	87	88	89	90	91	92	93	
1.....														
2.....	967	8.36	528	4.56	69	0.60	1,690	14.66						
3.....	1,049	9.06	540	4.67	117	1.01	1,677	14.49						
4.....	1,096	9.47	533	4.60	35	.30	1,363	11.78	14.60	6,853	499	5.11	8.69	
5.....	892	7.68	540	4.65	354	3.05	1,539	13.25	8.37	4,206	503	4.74	8.34	
6.....	1,033	8.92	556	4.80	457	3.95	1,509	13.03	9.55	5,494	575	4.44	7.58	
7.....	1,050	9.10	551	4.78	125	1.09	1,330	11.53	12.13	6,966	574	5.42	8.69	
8.....	1,127	9.71	560	4.82	106	.91	1,857	16.00	8.33	4,365	524	3.84	7.03	
9.....	1,123	9.69	549	4.74	211	1.82	1,325	11.44	9.87	5,651	573	5.34	8.92	
10.....	1,083	9.33	547	4.71	195	1.68	908	7.74	13.47	6,841	507	5.42	8.68	
11.....	1,116	9.66	567	4.91	142	1.23	669	5.78	9.12	5,824	639	4.75	7.51	
12.....	1,061	9.16	564	4.87	285	2.46	1,251	10.80	10.53	6,962	661	4.82	7.83	
13.....	1,017	8.76	556	4.79	276	2.38	1,068	9.37	9.30	5,496	591	5.16	7.50	
14.....	1,038	8.93	560	4.82	72	.62	1,211	10.42	12.22	7,252	594	5.32	8.50	
15.....	1,059	9.14	561	4.84	573	4.94	1,940	16.74						
15A.....	1,070	9.23	567	4.89	332	2.87	1,397	12.05						

^a Hydrogen and hydrocarbons not determined.^b Heat-balance items (designated by numbers in parentheses under this heading) are explained in Bull. U. S. Geol. Survey No. 325, p. 153.^c The per cent of heat absorbed by the boiler is shown under item 72*, column 67 of this table.

OBSERVERS' NOTES.

Density of smoke.—The visible smoke during all the tests was very light and appeared to be composed mostly of water vapor. At the top of the stack hardly any smoke was discernible, the color of the smoke becoming apparent only at a distance of 15 to 30 feet from the top of the stack, where a density was observed that would be denoted on Ringelmann's charts as between Nos. 0 and 1.

Test 1, October 8.—Test 1 was short and for that reason is not considered accurate. The fuel bed at the start was comparatively thin (about 6 inches), and when thickest was only 9 to 10 inches. Samples of coal and ash of this test were not sent for chemical analysis.

Test 2, October 10.—Test 2 was run with superheated steam in the ash-pit blowers. Fire was run at low rate of combustion two days before starting the test and the fuel bed was apparently in good condition when the test was started. The fuel bed at start was about 17 inches thick; average for the test about 20 inches. The grates were shaken eleven times during the test. The fire had burned down at the close to the same level as at the start. In computing results 650 pounds was added to the weight of coal as correction for the clinkers in the fuel bed at the end of the test. Fairly uniform conditions prevailed during the test.

Test 3, October 11.—Test 3 was run with superheated steam in the ash-pit blowers. Fire was cleaned about 1 or 2 o'clock in the morning and the boiler was run at 60 to 75 per cent of rated capacity until test started. The fuel bed was about 13 inches thick at the start and 18 inches at the close; average for the test about 20 inches. During the test about 165 pounds of coal was fired at intervals of two or three minutes, spreading the coal over the entire fuel bed. The grate was shaken ten times during test. Uniform conditions prevailed during the entire test.

Test 4, October 12.—Test 4 was run with superheated steam in the ash-pit blowers. The fuel bed was about 16 to 17 inches thick at the start and about 20 to 21 inches at the close; average during test about 20 inches. The grate was shaken eight times during the entire test. Coal was fired in charges of 160 pounds at intervals of four or five minutes, spreading it over the entire grate area. Uniform conditions prevailed during entire test. Clinkers were more troublesome than in tests 2 and 3. This was probably due to the fact that the coal contained about 30 per cent of slack. The fuel was apparently of inferior quality as compared with that of tests 2 and 3. Clinkers were heavy and fused to the brick on the sides of the furnace.

Test 5, October 14.—Test 5 was run with superheated steam in the ash-pit blowers. Coal used on this test was crushed by hand and contained very small percentage of slack; it was mostly in pieces 3 to 6 inches through. Sulphur appeared; distributed in thin layers. The fuel bed was about 15 inches thick at the start and about 18 inches at the close. The grate was shaken eight times during test. Fire was apparently in good condition during the entire test. Coal was fired in charges of 165 pounds at fifteen-minute intervals. From 3.30 to 5 o'clock the fan forcing the air into the furnace through the bridge-wall opening was out of repair and ran too slow, so that not enough air was supplied to burn the gases; this resulted in high CO content in the flue gases.

Test 6, October 16.—Test 6 was run with superheated steam in the ash-pit blower. Coal used on this test was crushed by hand and was wet from rain when fired. Sulphur appeared in rather large quantities; distributed in thin layers. The fuel bed was about 17 inches thick at the start and about 20 inches at the close; average during test about 22 inches. The grate was shaken five times during the test. Until about 3 o'clock the test was somewhat irregular; after that it ran smoothly. The grates were not shaken after 1.35 o'clock, and the fire remained in good condition. Coal was fired in charges of 165 pounds at four-minute intervals. The percentage of CO in the

flue gases, as shown by analysis, is high when the air pressure to bridge-wall openings is low; when the pressure is increased CO drops down.

Test 7, October 17.—Test 7 was run with superheated steam in the ash-pit blowers. Coal used was crushed by hand. Sulphur appeared in large quantity; distributed in thin layers. The fuel bed was 17 inches thick at the start and 22 inches at the close; average for the test about 23 inches. The grate was shaken four times during test. Little free ash was found in the ash pit at the end of the test. The grate seemed to be entirely covered by clinkers, and no more ash could be shaken down by rocking it. When pulled the clinkers had a very strong SO₂ odor. The fire was unmanageable during the last one and one-half hours, and the test could not have been run much longer.

Test 8, October 19.—Test 8 was run with saturated steam in the ash-pit blowers. Coal was crushed by crusher, was very wet, and contained 20 to 30 per cent of slack. The fuel bed was about 16 inches thick at the start and about 20 inches at the close; average for the test 18 to 20 inches. The grates were shaken five times during the test. Attempts to shake clinkers through the grate resulted in shaking a large amount of fine coal into the ash pit. In cleaning up after closing large and thick pieces of clinker were found on the grate.

Test 9, October 20.—Test 9 was run with saturated steam in the ash-pit blowers. Coal was crushed by crusher, was wet from rain, and contained about 20 per cent slack. The fuel bed was 15 inches thick at the start and 20 inches at the close; average for the test 20 to 22 inches. After 5 p. m. the fire seemed to be in bad condition on account of clinkers, preventing free passage of air through the fuel bed.

Test 10, October 21.—Test 10 was run with saturated steam in the ash-pit blowers. Coal was crushed by crusher. Fire was started at 3 a. m., with the burning coal left on the grate after cleaning. The fuel bed was 18 inches thick at the start and 22 inches at the close; average for the test about 20 inches. Coal was fired in charges of 165 pounds at five-minute intervals.

Test 11, October 22.—Test 11 was run with superheated steam in the ash-pit blowers. Coal was crushed by crusher; the appearance of the coal was good. The fuel bed was 14 inches thick at the start and 18 inches at the close; average for the test 20 inches. The grates were shaken eight times during the test. Strong SO₂ odor came out of the ash pit when grates were shaken. Coal was fired in charges of 165 pounds at three-minute intervals. This test represents the highest capacity which can be developed with this apparatus and fuel without great decrease of efficiency.

Test 12, October 23.—Test 12 was run with saturated steam in the ash-pit blowers. Coal was crushed by crusher. The fuel bed was 14 inches thick at the start and 18 inches at the close; average for the test 18 to 20 inches. The grates were shaken five times during test, only very little at a time, so that not much fine coal was shaken into the ash pit. Coal was fired in charges of 165 pounds at three-minute intervals.

Test 13, October 26.—Test 13 was run with superheated steam in the ash-pit blowers. Coal used had been exposed to wind and sun for forty-eight hours and was crushed by crusher. The fuel bed was 13 inches thick at the start and 17 inches at the close; average for the test about 21 inches. Coal was fired in charges of 165 pounds at intervals of four or five minutes. The grates were shaken six times during the test. One of the ash-pit blowers was stopped during the first three hours of the test.

Test 14, October 27.—Test 14 was run with superheated steam in the ash-pit blowers. Coal used had been exposed to sun and wind for twenty-four hours; the portion fired after 5.30 contained about 20 per cent of slack. The fuel bed was 13 inches thick at the start and 18 inches at the close; average for the test 21 inches. The grate was shaken four times during test. Coal was fired in charges of 165 pounds at four-minute intervals.

Test 15, October 29.—Test 15 was run with superheated steam in the ash-pit blowers, steam being supplied for this purpose by the main test boiler. One object of this test was to determine whether the clinkers could be made to fall into the ash pit by frequent

shaking of the grate and the fire thereby continually maintained in good running condition. At 9.45 a. m. one of the shaking bars on the left side of the grate became disconnected from the grate bars, and after that only the other (alternate) grate bars on the left side could be shaken. As a result of the frequent shaking of the grate much fine coal fell into the ash pit and the amount of refuse was increased considerably. At 6.50 p. m. the fire became unmanageable, on account of clinkers on the grate, and the test had to be stopped. The fuel bed was 18 inches thick at the start and 22 inches at the close; average for the test 22 inches.

Test 15 has been separated into two parts. The first four hours have been considered as preliminary. The intention was to run ten hours after that, but clinkers so accumulated on the grate that the second part of the test had to be stopped at the end of eight hours. The data of the two parts appear separately in Table 3. The whole test, including the preliminary part, is represented in the line opposite test number 15; the data of the last eight hours (omitting the preliminary four hours) are given in the line opposite 15A.

INTERRELATION OF DATA.

EFFECT OF VARYING RATES OF COMBUSTION.

Figures 4 and 5 show graphically the relation between the data taken during the tests and the calculated results. Each symbol on the chart represents one test. The hollow circles show tests made with saturated steam in the ash-pit blowers; solid circles show tests during which the ash-pit blowers were operated with superheated steam. The tests made with hand-crushed coal are represented by solid squares; coal on all other tests was crushed by a crusher. The two tests represented by triangles were made with coal weathered twenty-four and forty-eight hours.

Curve C of figure 4 shows the relation of the rates of combustion to equivalent evaporation per pound of dry coal. Although the symbols do not fall closely along a single line the general indication is that as the rate of combustion increases from 19 to 29 pounds of dry coal burned per square foot of grate per hour the equivalent evaporation per pound of dry coal falls from 6.4 to 5.8. This drop in evaporation when the rate of combustion increases is caused to some extent by less perfect absorption of heat by the boilers; but the principal cause, as will be shown in connection with figure 5, is less perfect combustion of the gaseous fuel. This curve also shows that hand-crushed coal does not give any better results than coal crushed by the crusher; it further shows that the use of superheated steam in the ash-pit blower does not improve the evaporation. The two tests made with the weathered coal show evaporation slightly above the average; the improvement, however, is rather small and hardly decisive. On examining the chemical composition of the coal of these two tests it is found that the weathering reduced the moisture but little, and that more could probably be gained by longer weathering.

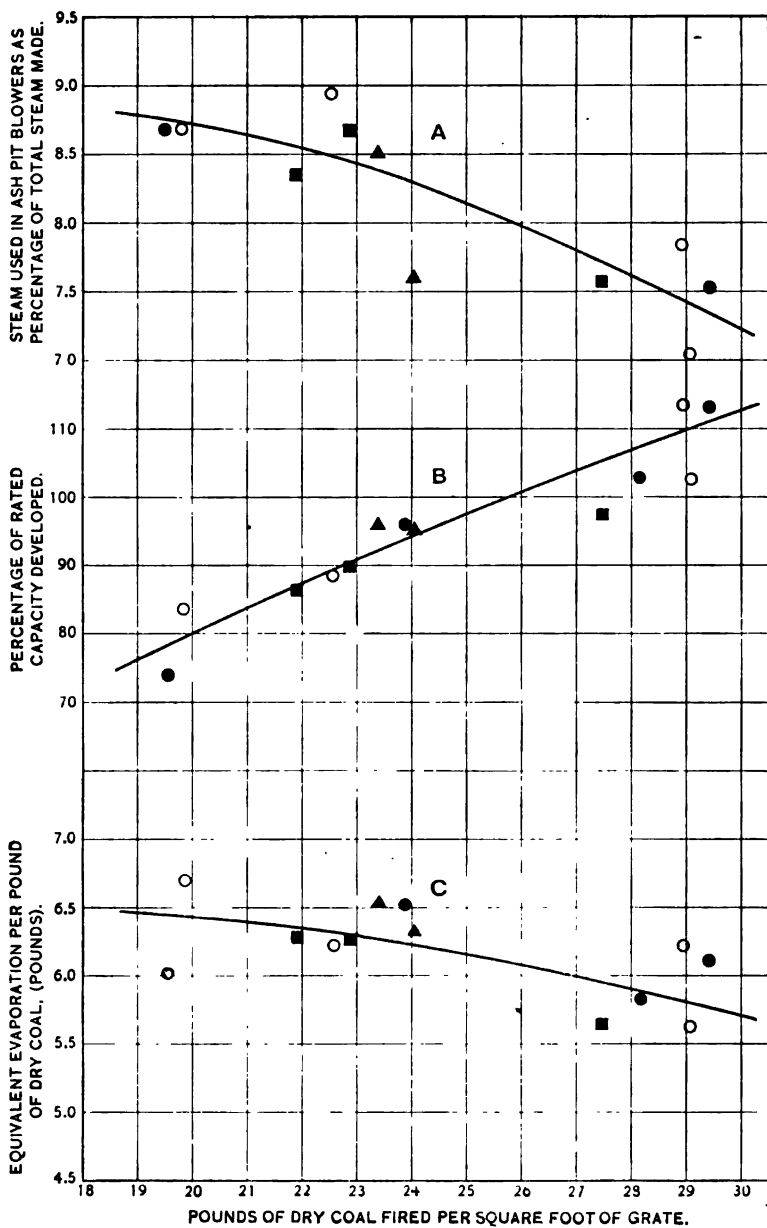


FIGURE 4.—Curves showing relations of rates of combustion to: A, Steam used in ash-pit blowers; B, percentage of boiler's rated capacity developed; C, equivalent evaporation per pound of dry coal fired.

Curve B of figure 4 shows the relation of the rates of combustion to capacity. The indication is that the capacity increases almost directly with the rate of combustion. This of course is reasonable, because when more coal is burned more heat is liberated in the furnace, and more heat is absorbed by the boiler.

Curve A representing the relation of the rate of combustion to the consumption of steam in ash-pit blowers shows that as the rate of combustion increases less steam is used. The actual drop in the consumption of steam is greater than the apparent as shown by the curve because of the fact that as the combustion increases the total steam made per pound of coal grows less, the basis of measuring the steam becomes smaller, and the consumption of the steam consequently appears larger than it really is. The curve indicates that as the rate of combustion increases from 19 to 29 pounds of coal per square foot of grate per hour the amount of steam used in the ash-pit blowers drops from 8.6 to 7.4 per cent of the total steam made. This drop is perhaps contrary to what would be ordinarily expected. In fan blowers the steam consumed would increase approximately as the square of the rate of combustion. The smaller consumption of steam by the Argand blowers at higher rates of combustion is undoubtedly due to a much increased efficiency of the blower at the higher speed. To get higher rate of combustion more air must be supplied to the ash pit; this is attained by increasing the steam pressure immediately back of the small orifices of the blower. The increased steam pressure causes the steam discharged through the orifices to move at higher velocity, resulting in an increased air supply. As this higher velocity of steam makes the blower more efficient, it seems that the blower could be made more efficient at lower speed by making each orifice in the blower smaller; this change would retain higher pressure back of the orifice and thus increase the velocity of steam issuing from it. The objection to small orifices is that they are liable to become clogged, particularly if saturated steam is used. Perhaps fewer openings and those of larger size would increase the efficiency of the blowers without giving serious trouble on account of the openings becoming clogged. It may be noted that the amount of steam used during a given period in the blowers is too high and that considering the already great abundance of moisture in the coal steam is not needed in the ash pit. This suggests the possibility that the same results could be accomplished with much less expenditure of steam by the use of a fan blower.

Curve A shows that the consumption of steam in the blower is about the same whether saturated or superheated steam is used. There seems to be no advantage in the use of superheated steam other than that of lessening the chances of clogging the lower openings.

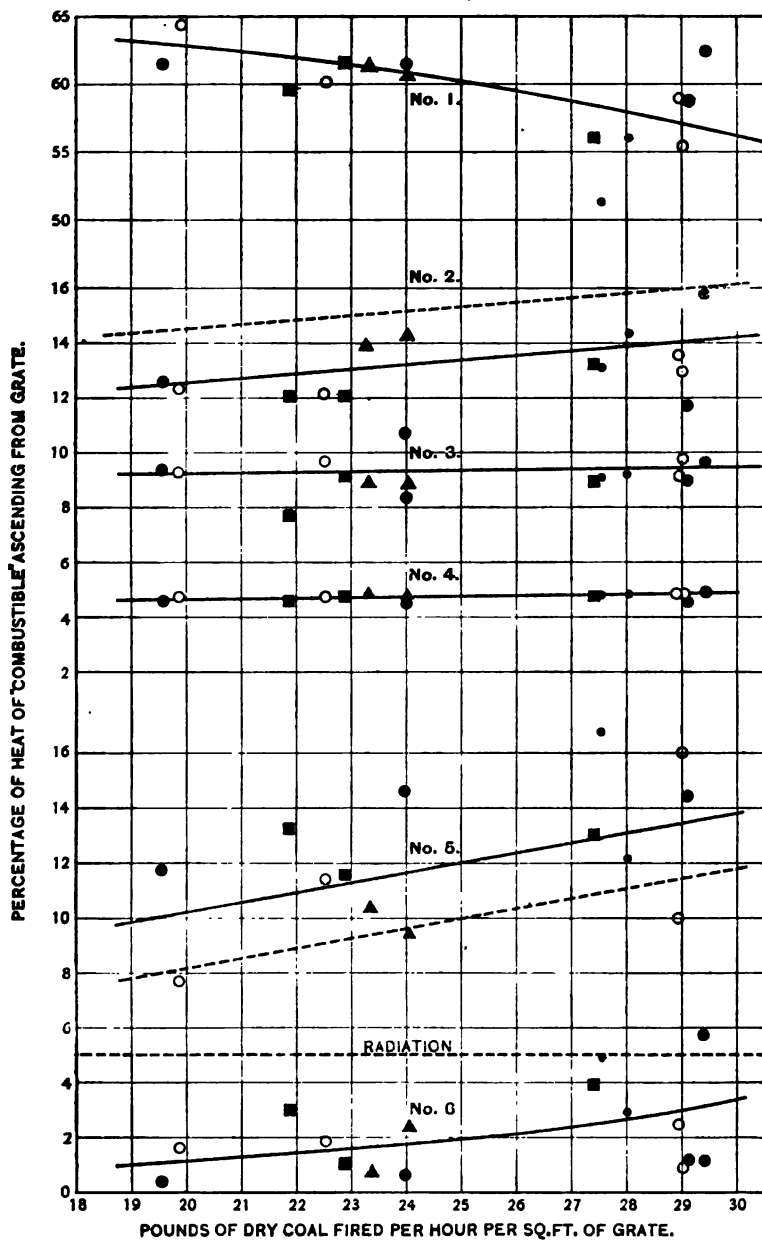


FIGURE 5.—Curves showing relations of rates of combustion to: 1, Heat absorbed by boiler; 2, heat lost in dry chimney gases; 3, heat lost in moisture in coal; 4, heat lost in moisture formed by the burning of hydrogen of coal; 5, heat unaccounted for; 6, heat lost in unburned CO in the flue gases.

Figure 5 shows the variation in the distribution of heat as the rate of combustion increases. Each item of the heat distribution is expressed in percentage of the total heat of combustible ascending from the grate. The amount of combustible thus consumed (column 23 of Table 3) is equal to the total combustible of coal as fired minus the amount of combustible in the refuse. The following equation shows the method of computing this factor (code item 30*):

$$\left\{ \begin{array}{l} \text{Com-} \\ \text{bustible} \\ \text{ascending} \\ \text{from grate} \\ \text{(pounds)} \end{array} \right\} = \left\{ \begin{array}{l} \text{Total} \\ \text{coal as} \\ \text{fired} \\ \text{(pounds)} \end{array} \right\} \frac{100 - \left\{ \begin{array}{l} \text{Moisture} \\ \text{in coal} \\ \text{(per cent)} \end{array} \right\} - \left\{ \begin{array}{l} \text{Ash in} \\ \text{coal} \\ \text{(per cent)} \end{array} \right\}}{100} - \left\{ \begin{array}{l} \text{Total} \\ \text{refuse} \\ \text{(pounds)} \end{array} \right\} \frac{\left\{ \begin{array}{l} \text{Com-} \\ \text{bustible} \\ \text{in refuse} \\ \text{(per cent)} \end{array} \right\}}{100}$$

The curves of figure 5 represent the items of the heat balance (columns 78-88, Table 3) plotted on the rates of combustion as abscissas. The ordinates to be used with each curve are directly at the left.

Curve 1 shows the relation of the rate of combustion to the amount of heat absorbed by the boiler which constitutes item 1 of the heat balance (column 78, Table 3), stated in percentage form as item 72* (column 67) in Table 3. In the code of the American Society of Mechanical Engineers for conducting steaming tests this item is designated "boiler efficiency," but strictly speaking it is the combined efficiency of the boiler and furnace exclusive of the grate.

The curve shows that in general this efficiency drops from about 63 to 57 per cent as the rate of combustion increases from 19 to 29 pounds of dry coal per square foot of grate per hour. This drop of about 6 per cent in efficiency is perhaps mostly due to less complete combustion of the gaseous combustible in the combustion space of the furnace. This statement is supported by the showing of curves 5 and 6. Curve 6 shows the heat loss caused by carbon monoxide (CO) leaving the furnace unburned. The indication is that as the rate of combustion increases from 19 to 29 pounds, the loss in CO increases from 1 to 3 per cent. When CO leaves the furnace unburned there is a very strong probability that traces of other combustible gases, such as H_2 , CH_4 , C_2H_4 , and C_2H_2 , are escaping unburned; also that small quantities of tar vapors and perhaps some solid carbon in the form of fine soot are passing unconsumed out of the furnace and causing further losses. Although each of these combustibles may escape in amounts so small that it is very difficult or impossible to measure them, there are probably so many of them that the total heat loss due to their leaving the furnace unburned may be much higher than the loss due to CO. This loss due to the unconsumed hydrogen and carbon and their various combinations appears in item 6 of the heat balance (columns 87 and 88, Table 3), which item

is plotted as curve 5 of figure 6. This item is called the unaccounted-for heat loss, and includes, besides the loss mentioned as due to incomplete combustion, also the loss due to radiation, as well as all the errors of observation, of sampling of coal and ash, and of chemical analysis. These errors are not likely to vary with the rate of combustion. Radiation varies with temperature only and not with the rate of making steam; therefore, the radiation loss decreases rather than increases with the rate of combustion. We may reasonably infer that if the unaccounted-for loss increases with the rate of combustion, the increase is due to the unconsumed tar vapors and gases as suggested above. On examining curve 5 it is seen that on the average the unaccounted-for loss rises from 10 to 13.5, or 3.5 per cent, as the rate of combustion increases from 19 to 29 pounds. This 3.5 per cent loss plus the 2 per cent increase in loss due to CO may perhaps account for 5.5 per cent of the total drop of 6 per cent in the efficiency accompanying the increased rate of combustion. These losses amounting to 5.5 per cent are caused by incomplete combustion and should be charged against the furnace.

Curve 2, figure 5, represents item 4 in the heat balance (columns 79 and 80, Table 3) and shows the relation of the rates of combustion to the heat carried away in dry chimney gases. The curve indicates that this loss increases from 12.5 to 14, or 1.5 per cent, as the rate of combustion increases from 19 to 29 pounds. This increase of heat carried away in dry chimney gases is caused by the rise of the temperature of the chimney gases; in other words, it is due to less complete absorption of the heat by the boiler from the gases. The increase in this loss is therefore directly chargeable to the boiler.

Curve 3 represents item 2 of the heat balance (columns 81 and 82, Table 3) and shows the relation of the rates of combustion to the heat carried away in moisture in coal. The symbols fall very closely along the line of the curve, showing that there was very little variation of this loss on different tests. Very little variation is due to the rate of combustion; the curve shows that the loss in the moisture in coal increases only about 0.25 per cent, while the rate of combustion ranges from 19 to 29 pounds.

Curve 4 represents item 3 of the heat balance (columns 83 and 84, Table 3) and shows the heat loss in moisture formed by burning the hydrogen of the coal. It has been plotted rather to complete the graphical heat balance of this figure than to show any relation to other items. All the symbols of this item fall very closely along the curve, showing that this loss is about the same for all tests and very nearly the same for all rates of combustion.

It has been stated on page 15 that the recorded flue-gas temperature was probably 50° F. lower than the actual temperature. Assuming the correctness of this estimate of variation from the actual, the heat loss in dry chimney gases should be recorded 2 per cent higher

and the unaccounted-for loss 2 per cent lower. Were the flue-gas temperature 50° F. higher, curve 2 of figure 5 should be shifted to the position of the dotted line above it and curve 5 to the position of the dotted line below it, but the relative position of the symbols along the curves would not be changed. The unaccounted-for loss would then run from 8 to 11.5 per cent, which is probably what it actually was.

It has been pointed out that for the investigated increase of the rate of combustion the heat loss in CO has increased 2 per cent and the unaccounted-for loss has increased 3.5; the probability has also been pointed out that the increase in the unaccounted-for loss was due to unconsumed combustible gases and tar vapors. Assuming the radiation loss to be 5 per cent and constant for all rates of combustion, by subtracting this 5 per cent from the unaccounted-for loss we have a remainder showing the loss due to unconsumed hydrogen and hydrocarbons to be represented by the corrected flue-gas temperature curve. This remainder is 3 per cent at the 19-pound rate of combustion and 6.5 per cent at the 29-pound rate of combustion. The increase is about the same as is shown by the CO curve. It should be stated here that CO in the flue gases should be considered as an indication of other losses due to incomplete combustion.

Steaming tests made by the Geological Survey with other coals have shown the same relation between the CO loss and the unaccounted-for loss, indicating that these two losses rise and fall together and that the variable part of the unaccounted-for loss is the loss due to incomplete combustion.

The showing of figure 5 may be summed up as follows: As the rates of combustion increase from 19 to 29 pounds the efficiency drops from 63 to 57 per cent, a drop of 6 per cent. About 5 per cent of this drop can be accounted for by incomplete combustion as shown by curves 5 and 6, and 1 per cent by incomplete absorption of heat by the boiler. In other words, of the total drop in efficiency about 5 per cent is probably due to lower furnace efficiency and 1 per cent to lower true boiler efficiency.

EFFECT OF MOISTURE IN COAL.

The coals burned during the tests reported in this bulletin contained very nearly the same percentage of moisture, so that the effect of moisture in coal on the economy of the operation of the steam-generating apparatus can not be deduced from the results of these tests. It is known that the presence of a small percentage of moisture may help the combustion enough to neutralize its tendency to reduce, by its high latent and specific heat, the temperature of the products of combustion; but if the moisture exceeds certain limits the reduction in temperature outweighs the advantages in combustion which may possibly be gained by its presence, and the economic

efficiency of operation of a steam plant is lowered with every increase of moisture in the coal. Just what percentage of moisture in coal is beneficial in burning the coal under a boiler is at the present state of knowledge on this subject impossible to say. It undoubtedly varies with different coals. The treatment of this subject in this report relates only to the cooling effect of the moisture in coal on the products of combustion.

Heat flows of its own accord only from hot bodies to cold bodies; therefore only that portion of the heat in the products of combustion which is above the temperature of the water in the boiler can be absorbed by the water. That portion of the heat in the products of combustion which is below the temperature of the water in the boiler is not available for making steam. It follows that such condition of coal and such methods of burning it as will evolve the maximum amount of the heat in coal at a temperature above that of steam and a minimum of heat below that temperature will give the best results under a steam boiler. The harmful effect of excessive moisture in coal on steam-boiler operation is due to the fact that a large part of the heat held by the moisture is below the temperature of the water in the boiler and hence is not available for making steam. This can be illustrated by a specific example.

Suppose a boiler operates under a pressure corresponding to steam temperature of 350° F. (about 135 pounds, absolute), that the temperature of the product of combustion is $2,000^{\circ}$ F., and that the temperature of the atmosphere is 70° F. Under these conditions 1 pound of moisture in the products of combustion contains heat units below the temperature of the water in the boiler (and hence not available for absorption by the boiler), as follows: Below the boiling point (212° – 70°), 142 B. t. u.; latent heat of steam, 965 B. t. u.; superheating steam to 350° F. (350° – 212°) 0.48, 66 B. t. u.—making a total of 1,173 B. t. u. The heat units in this pound of moisture above the temperature of the water in the boiler (and hence available for absorption) are ($2,000^{\circ}$ – 350°) 0.48 = 792 B. t. u.

These figures show that the larger part of the heat in the moisture is not available for making steam. In fact, the moisture in coal and the moisture formed by burning the hydrogen of coal are equivalent to a small boiler placed in the furnace and making steam equal in weight to the moisture passing out of the furnace, thus absorbing that amount of heat before the main boiler gets a chance to absorb it.

In coals having high moisture content, such as the lignite used during the tests reported in this bulletin, the heat required to evaporate the moisture is taken from the highest temperature of the products of combustion, thereby reducing the heat available for absorption by the boiler and thus reducing the useful effect of the coal. This is shown in figure 6, illustrating the results of test 4, in burning coal with varying supplies of air. The coal used during test 4 contained 44.26 per

cent of moisture. Calculations were made on the basis of 1 pound of combustible; moisture formed by the burning of hydrogen was added to the moisture in coal. Specific heat of water was taken as 1; latent heat of steam as 965; specific heat of steam as 0.48, constant at all temperatures; and specific heat of the gaseous products of combustion as 0.24, constant at all temperatures. The heat value of 1 pound of combustible, determined by calorimeter, was 11,567. The same calculations were made for coal with the moisture reduced to 20 per cent and for coal chemically dry. In all computations it was assumed that the combustible was completely burned, a condition which in practice would be difficult to obtain, especially with the low supply of air. The curves end at a point where the air supply is 10 pounds, which is near the theoretical minimum air supply with sufficient oxygen for complete combustion.

Curves 1, 2, and 3 should be studied in connection with the scales at the left and at the bottom of the figure. They show what theoretically maximum temperatures can be obtained with each air supply. Curve 3 gives the temperatures for coal having 44.26 moisture; curve 2 gives the temperature of the coal with the moisture reduced to 20 per cent; curve 1 gives the temperature of the coal containing no moisture at all. Each of these curves shows that the temperature rises rapidly as the air supply is reduced. In practice this rise would not be so rapid, because as the air supply is reduced the combustion would be less complete and only part of the heat in the coal would be liberated; the temperature of the products of combustion would consequently be more or less lower than that indicated by the curves in the figure, the completeness of combustion depending on the fitness of the furnace to burn this kind of coal and on its manipulation. The three curves collectively show that for each 20 per cent reduction in moisture the temperature of the products of combustion with the same supply of air rises from 200° to 500° F.

Curves 4, 5, and 6 of figure 6 should be studied in connection with the upper scale at the right and the scale at the bottom of the figure. These curves show how much of the heat is above temperature 350° F., when 1 pound of combustible is burned with varying air supply, indicating the amount of heat available for a boiler operating under a water temperature of 350° F. Curve 6 shows this relation for the coal containing 44.26 per cent moisture; curve 5 for coal with the moisture reduced to 20 per cent, and curve 4 for coal with no moisture. The amount of gain in the available heat when the moisture is reduced is shown by curves 7 and 8, which together with curve 9 should be studied in connection with the lower scale at the right and the scale at the bottom of the figure. Curve 8 shows the higher percentage of the total heat of the combustible which is made available for the boiler by reducing the moisture from 44 to 20 per cent. Curve 7 shows how much more heat could be made available with the coal chemically dry.

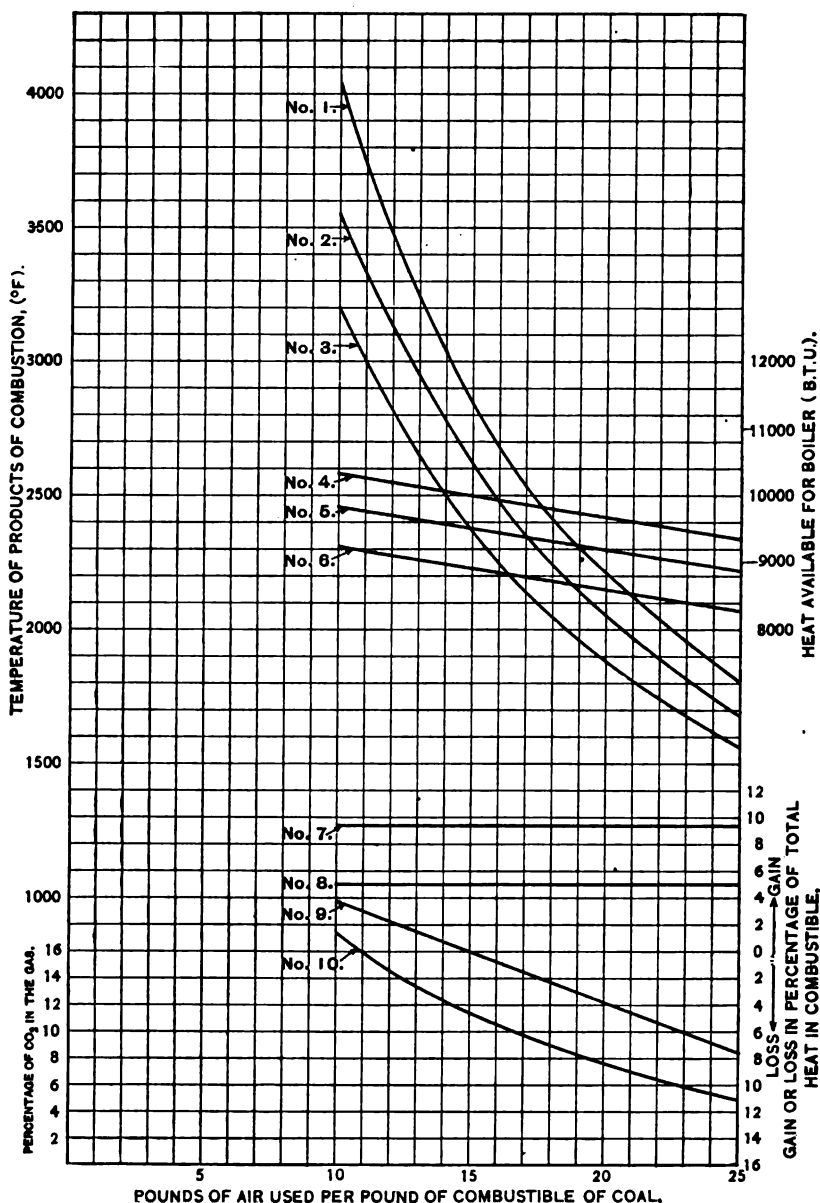


FIGURE 6.—Curves showing relation of the temperature of the products of combustion to the weight of air used to burn 1 pound of combustible; 1, for lignite chemically dry; 2, for lignite containing 20 per cent moisture; 3, for lignite containing 44 per cent moisture. Heat available for boiler in burning 1 pound of combustible of lignite: Curve 4, with moisture previously driven off; curve 5, with moisture reduced to 20 per cent; curve 6, with 44 per cent moisture. Curve 7 shows gain of conditions of curve 4 over conditions of curve 6. Curve 8 shows gain of conditions of curve 5 over conditions of curve 6. Curve 9 shows gain or loss when less or more than 15 pounds of air is used to burn 1 pound of combustible. Curve 10 shows percentage of CO₂ in the flue gases under varying air supply.

The curves show that by driving off 24 per cent of the moisture before burning the coal 5 per cent more of the total heat of combustible could be made available to the boiler, and that if the moisture were entirely driven off before burning the coal the heat available for the boiler would be increased by 9.5 per cent.

All the tests reported in this bulletin were run with about 15 pounds of air per pound of combustible. Curve 9 indicates how much more heat could be made available if the air supply were reduced below 15 pounds, provided the completeness of combustion would not be lowered. The curve also indicates the decrease in the heat available if the air supply were increased beyond 15 pounds. The curve passes through zero point at the air supply equal to 15 pounds; increase in heat available for boiler is shown above the zero line, decrease in available heat below the line. It is shown that by decreasing the air supply to almost the theoretical minimum only about 3.5 per cent more heat could be made available for the boiler. It is doubtful whether in practice anything could be gained by reducing the air supply much below 15 pounds of air, because more heat would probably be lost by the incomplete combustion resulting from insufficient supply of oxygen.

Curve 10 should be studied in connection with the lower scale at the left and the scale at the bottom of the figure. The curve shows the approximate percentage of CO_2 in the flue gases under varying air supply. Thus when 15 pounds of air per pound of combustible are used the flue-gas analysis will show about 11 per cent of CO_2 .

Figure 7 shows the effect of moisture in coal on the temperature of the products of combustion, the heat available for boiler, and the heat not available for boiler, when coal is burned with a constant supply of 15 pounds of air per pound of combustible, assuming that all the combustible burns perfectly.

Curve 1, to which the scales at the right and bottom of the figure apply, shows the relation of the percentage of heat available for the boiler to the percentage of moisture in coal. Thus, when the moisture in coal is 44 per cent, only 76 per cent of the total heat of coal can possibly be made available for the boiler; at 0 moisture the available heat rises to 86.5 per cent.

Curve 2, to which the scales at the left and bottom of the figure apply, shows the relation of the temperature of the products of combustion to the percentage of moisture in coal. The indication is that as the moisture decreases from 44 per cent to 0 the temperature of the products of combustion rises from $2,350^\circ$ to $2,850^\circ$ F.

Curves 3, 4, and 5 should be interpreted by the scale at the right. Curve 3 gives the total percentage of heat in the products of combustion unavailable for the boiler; that is, the heat which is below the temperature of the water in the boiler. Curve 5 gives the percentage of unavailable heat which is in the dry gaseous product of

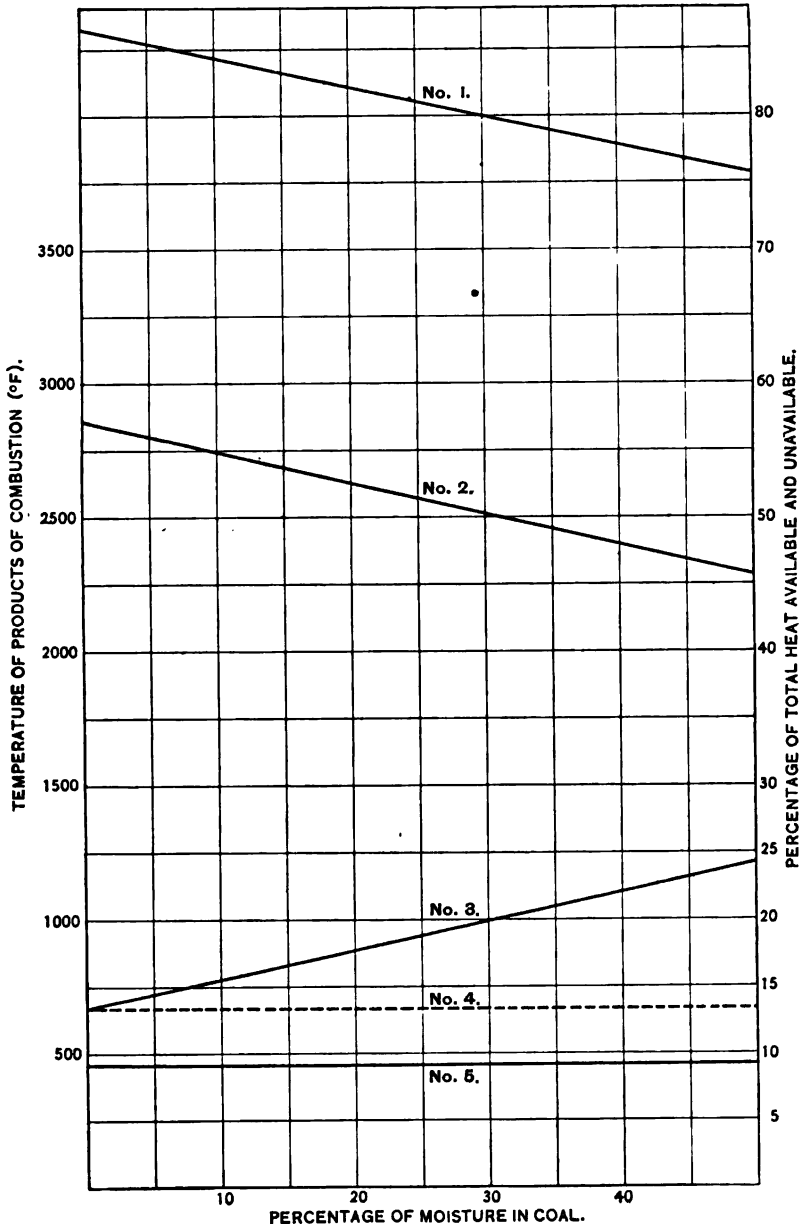


FIGURE 7.—Curves showing effect of moisture when 15 pounds of air are used to burn 1 pound of combustible: 1, on the percentage of heat available for boiler; 2, on temperature of products of combustion; 3, on the percentage of heat unavailable for boiler; 4, on the percentage of heat unavailable (curve 3) less that held by the free moisture in coal; 5, on the percentage of heat that is unavailable and is contained in dry gases. The difference between the ordinates of curves 4 and 5 gives the unavailable heat held in the moisture formed by the burning of the hydrogen of coal.

combustion. As the computation for figure 7 is made for a constant 15-pound supply of air, the percentage of heat unavailable in the gases is constant for all moistures. The difference between the ordinates of curves 5 and 4 gives the unavailable heat held in the moisture formed by the burning of the hydrogen of coal. The difference between the ordinates of curves 4 and 3 gives the unavailable heat held by the free moisture in coal. It is apparent from the figure that at the high-moisture end much more unavailable heat inheres in the combined moistures than in the dry products of combustion.

Table 4 gives the calculated values of heat available and unavailable for the boiler when burning the coal of test 4 with four different supplies of air and also four different moisture contents. The amount of heat is expressed in percentage of the total heat in the coal. Heat above 350° F. is taken as available whether in dry gases or in moisture; the unavailable is the heat below this temperature. The unavailable heat is separated into two parts, the unavailable heat in dry gases and that in the moisture, the latter being the sum of the moisture in coal and the moisture formed by burning the hydrogen of the coal. In the table each vertical column gives the percentages of heat for the same moisture in coal, but with different air supply. The last column under the heading "no moisture," shows that the unavailable heat in moisture is 4.4 per cent, which is the heat in the moisture formed by the burning of the hydrogen of coal; this moisture can not be avoided under any circumstances. The table shows clearly that so far as unavailable heat is concerned 44 per cent of moisture in coal, in addition to the moisture formed by burning hydrogen, is more harmful than 20 pounds of dry gas; also that more heat is made available in using 15 pounds of air per pound of combustible when moisture is reduced to 20 per cent, than in using the theoretical minimum supply of air when the moisture is kept at 44 per cent.

TABLE 4.—*Calculated values of heat in coal^a available and unavailable for boiler under varying conditions of air supply and moisture.*

Weight of air used to burn 1 pound of combust- ible (pounds).	Moisture in coal (per cent).											
	44			20			10			No moisture.		
	Heat avail- able for boiler.	Unavailable heat in—		Heat avail- able for boiler.	Unavailable heat in—		Heat avail- able for boiler.	Unavailable heat in—		Heat avail- able for boiler.	Unavailable heat in—	
		Dry gases.	Moist- ure.		Dry gases.	Moist- ure.		Gases.	Moist- ure.		Gases.	Moist- ure.
10.....	79.9	6.3	13.8	85.0	6.3	8.7	87.1	6.3	6.6	89.3	6.3	4.4
15.....	77.0	9.2	13.8	82.1	9.2	8.7	84.2	9.2	6.6	86.4	9.2	4.4
20.....	74.1	12.1	13.8	79.2	12.1	8.7	81.3	12.1	6.6	83.5	12.1	4.4
25.....	71.2	15.0	13.8	76.3	15.0	8.7	78.4	15.0	6.6	80.6	15.0	4.4

^a The amount of heat is expressed as a percentage of the total heat in the coal.

On test 4 the coal was burned with approximately 15 pounds of air per pound of combustible, and it contained 44.26 per cent of moisture. Table 4 shows that if all the combustible was burned perfectly only 77 per cent of the total heat in the coal could possibly be made available for the boiler. The gas analysis and the heat balance show, however, that the combustible was not burned perfectly and that about 5 or 6 per cent of the heat in coal was not developed in the furnace. This loss through incomplete combustion reduces the heat available to boiler to about 71 or 72 per cent. Furthermore, part of the heat developed in the furnace was radiated away through the walls and especially through the numerous cast-iron doors in the furnace and boiler setting. Probably a fair estimate of this loss by radiation is about 3 per cent. Deducting this loss from the amount of heat available leaves only about 69 per cent of the total heat in the coal which is really available for the boiler. No boiler ever built absorbs all the heat that is available for absorption; that is, no boiler ever reduces the temperature of the furnace gases to the temperature of the steam in the boiler. In the case of the boiler under consideration, the gases left the heating surface from 200° to 300° F. higher than the temperature of the steam in the boiler. Assuming that the true boiler efficiency of this particular boiler was 90 per cent, which estimate is rather high, the heat actually absorbed by the boiler is reduced to $69 \times 0.90 = 62.1$ per cent. Further loss of heat after being absorbed occurs by radiation from the boiler proper through the heads of the drum and through the thin brick covering on top of the steam drum. If this loss is assumed to be 2 per cent the heat carried away in steam would be reduced to 60 per cent of the heat in the coal. This is about the average actually obtained during the present series of tests.

Even if the incompleteness of combustion were reduced to, say, 2 per cent, which would be considered an unusually good result, the corresponding figures would be as follows: Heat available as generated in furnace, $77 - 2 = 75$ per cent; heat actually available after radiation loss from furnace, $75 - 3 = 72$ per cent; heat absorbed by boiler, $72 \times 0.90 = 64.8$ per cent; heat carried away in steam, $64.8 - 2 = 63$ (about) per cent.

Even should the air supply be reduced to nearly its theoretical minimum, the efficiency could not be raised very much. Thus, supposing it were possible to burn the coal with 10 pounds of air per pound of combustible, and estimating the loss through incomplete combustion at only 2 per cent, the corresponding CO₂ content of the flue gases would be about 17 per cent. For these conditions the next efficiency would be: Calculated maximum available heat (Table 4), 79.9 per cent; heat actually made available in furnace, $79.9 - 2 = 77.9$ per cent; heat actually available for boiler after radiation loss, $77.9 - 3 = 74.9$ per cent; heat absorbed by boiler, $74.9 \times 0.90 = 67.4$ per cent; heat carried away in steam, $67.4 - 2 = 65.4$ per cent.

These calculations are only approximate, but they show what can be expected from this coal when used for steaming purposes.

The above calculated figures expressing the heat carried away in steam should be compared with the results actually obtained and given in column 68 (72*) of Table 3. The tabulated figures include the incomplete combustion in the furnace but do not include the combustible lost through the grate. The amount of loss through the grate depends on how often and how much the grates are shaken, and may amount to several per cent. Upon comparison of the calculated results with those of Table 3, it must be admitted that the results obtained during the tests herein reported are very good indeed; in fact they are as good as those obtained in many plants using the best of coal.

COMPOSITION OF FLUE GAS.

COMPARISON OF SAMPLES.

Table 5 shows the chemical composition of flue gas, the samples of which were collected at two different places in the boiler setting. The left-hand half of the table gives the composition of gas collected through the regular flue-gas sampler (described on p. 10) which was located in the uptake. The right-hand half of the table gives the composition of gases collected through a single one-fourth-inch pipe inserted through a hole in the lowest middle door in the rear of the boiler setting. The end of this single sampling pipe extended into the stream of the gases at the turn from the second to the third nest of boiler tubes. The object of taking gas samples in this position was to get samples that would be comparable with samples taken by the engineers of the plant in tests made the previous year, during which the same boiler was used.

TABLE 5.—Comparison of composition of flue gas sampled in two different positions.

No. of test.	Sampler in the uptake.					Sampler at middle door in rear of boiler.				
	Time of collecting sample.	Constituent gases (per cent.).				Time of collecting sample.	Constituent gases (per cent.).			
		CO ₂ .	O ₂ .	CO.	N ₂ .		CO ₂ .	O ₂ .	CO.	N ₂ .
11	3.15-3.45...	11.4	8.2	0	80.4	3.30-3.31...	13.6	6.0	0	80.4
	4.15-4.45...	10.8	8.4	.3	80.8	4.28-4.29...	17.8	.8	1.2	80.2
	5.00-5.20...	9.8	9.6	0	80.6	5.15-5.16...	13.8	5.6	0	80.6
	8.30-8.50...	10.6	8.0	1	80.4	8.40-9.00...	15.6	3.2	0	81.2
	9.30-10.00...	12	6.6	.8	80.6	9.40-9.55...	16.8	.6	1.6	81.0
12	10.30-11.00...	12.2	5.2	2.2	80.4	10.50-11.00...	15.4	.2	3.4	81.0
	2.00-2.30...	11.6	7.2	0	81.2	2.00-2.30...	17	1.2	0	81.8
	3.00-3.30...	12.4	6.2	0	81.4	3.00-3.30...	17.6	.4	0	82.0
	4.05-4.25...	12.6	6.0	.2	81.2	4.05-4.25...	17.8	.6	.2	81.4
	11.05-11.30...	11.4	7.8	0	80.8	11.05-11.30...	15.6	3.0	0	81.4
14	1.00-1.20...	10.2	9.4	0	80.4	1.00-1.20...	14.8	4.6	0	80.6
	2.00-2.25...	12.8	6.6	.2	80.4	2.05-2.30...	17.4	1.0	1.0	80.6
	4.00-4.30...	12.8	6.0	.8	80.4	4.05-4.30...	17.4	.6	1.6	80.4
	7.30-8.00...	10.2	9.0	.2	80.6	7.30-8.00...	16.0	2.8	.4	80.8

The table shows that the sample collected through the single pipe ran 2 to 6 per cent higher in CO₂ and 2 to 6 per cent lower in O₂,

than the sample collected through the regular sampler placed in the uptake. The CO content is also higher in the right-hand half of the table than in the left. There are two causes for the difference in composition of the two samples.

(1) Air leaks into the setting and increases the percentage of O_2 , decreasing CO_2 by nearly the same amount. The leakage is particularly large in the last part of the gas passage on account of the numerous accessory doors and the preheating coils. There are also small but numerous openings around all metal objects which are bricked into the wall of the setting, because of the difference of contraction and expansion of metal and brick wall in response to thermal changes, and the contraction of the wall due to the drying mortar. This leakage is especially large in the last part of the gas passage because of the pressure drops from the outside air into the setting; or, in other words, because "the suction draft" is here larger than in any other part of the boiler setting.

(2) The second cause of the difference in gas composition is that the single-tube sampler drew the sample only from the central stream of gases, which was particularly high in CO_2 . The sample thus taken did not represent the average gas composition of the entire cross section of the gas passage. Undoubtedly if the gas sample taken in the same cross section had been taken near the walls, it would have shown that the percentage of CO_2 at that place is even lower than the one taken in the uptake. On account of the construction and location of the sampler in the uptake the sample taken with it represented very nearly the average gas composition of that particular cross section. As the gas analysis was to be used in the computation of the heat carried up the stack, the sample should be taken just as the gases leave the heating plates of the boiler and air preheaters. Therefore the sampler was properly placed as it was. On the whole, the gas composition near the stack is good, except that occasionally, perhaps, the CO runs too high for good economy.

GAS ANALYSIS AS A HELP IN OPERATING.

The semigas-producer furnace with which these tests were made is rather delicate in its operation. Sometimes the least change of the air supply through the bridge wall or the air pressure in the ash pit, or even the condition of the fuel bed may result in high CO in the flue gases, indicating incompleteness of combustion. On a dry, clear day this incompleteness of combustion might be detected by a careful observer in the brownish color of the smoke at the top of the stack. On a damp, cloudy day, however, the moisture in the chimney gases condenses at the top or even in the stack; the fog or mist thus formed changes the color of the smoke and conceals the conditions within the furnace. The best method of learning the furnace conditions from time to time is to take a sample of gas and

analyze it. The air supply through the bridge wall and the pressure in the ash pit may be regulated as need is shown by the analysis.

The function of the fuel bed is to generate the combustible gas; the function of the space above and beyond the bridge wall opening is to burn this combustible gas after air has been added. Now, if the gas analysis shows that a considerable quantity of CO escapes unburned, it is because too much gas is generated in the fuel bed for the amount of air supplied through the bridge wall. The remedy is either to increase the air supply by increasing the pressure in the duct bringing the air to the bridge wall openings or to reduce the rate of gas generation in the fuel bed by reducing the pressure in the ash pit; or both methods may be used together. Again, if the gas analysis shows high O₂ and no CO, too much air is used to burn the gases, the result being that the temperature of the products of combustion is low and less heat is available for the boiler. The remedy is either to reduce the air supply by decreasing the pressure in the air duct to the bridge wall or to generate more gas by increasing the pressure in the ash pit. Under varying loads the two pressures also have to be varied in order that the rate of heat generation in the furnace may correspond to the boiler's demand for steam.

It is impossible to determine in advance for a given capacity or rate of making steam the pressures required in the air duct to the bridge wall and in the ash pit to secure the best results, because the resistance of the fuel bed varies in accordance with the amount of ash and clinker accumulated therein. With low resistance to the passage of air through the fuel bed lower pressure in the ash pit will generate gas at the same rate as higher air pressure when the resistance of the fuel bed is high.

The resistance of the fuel bed will also vary with the size of the coal fired; smaller sizes of coal offer higher resistance to the passage of air than is offered by large uniform sizes.^a

FORMATION OF CLINKER IN THE FUEL BED.

The ash of the coal fused at a comparatively low temperature, so that clinker was easily formed in the fuel bed. In selecting this furnace to burn this particular coal it was hoped that the solid combustible of the coal could be changed into gaseous combustible at low fuel-bed temperature, so as not to fuse the ash, and that the gas could then be burned in the combustion space at high temperature. But solid fuel to be gasified must be heated. Within certain limits the richness of the gas depends on the temperature—the higher the temperature the richer the gas.^b

^a See Ray, W. T., and Krelsinger, Henry, Significance of drafts in steam boiler practice, Bull. U. S. Geol. Survey No. 367.

^b See Clement, J. K., Adams, L. H., and Haskins, C. H., essential factors in the formation of producer gas, Bull. Bureau of Mines No. 7. See also Temperature differences in fuel bed, by J. K. Clement and H. A. Grine, in Bull. U. S. Geol. Survey No. 393, pp. 15-26.

Even the moderate temperature at which this furnace is operated is sufficient to fuse the ash and form clinker. Generally the clinker begins to form along the walls of the furnace where enough air passes along the wall to burn the fuel completely instead of only to gasify it, thus developing a temperature sufficiently high to fuse the ash. From the walls the clinker grows toward the center of the grate until the grate is entirely covered with a layer of clinkers. Shaking the grate usually helps but little; the pieces of clinker are large enough to extend over two or three grate bars, and when the bars are shaken the clinker simply rides on top of the bars and is seldom caught between them and made to fall into the ash pit. When starting to form the clinker is usually fused to the wall and therefore can not be shaken through the grate. When the grate is shaken there is always danger of shaking too much burning coal into the ash pit. The coal lies in very small pieces between and on top of the pieces of clinker, and is easily sifted through the grate into the ash pit, where it burns and heats the grate and the fuel bed above, thus helping to form more clinker. Apparently the best way to rid the fuel bed of clinkers would be to provide doors in the front of the furnace low enough so that the clinker could be easily removed by a hook or a hoe. This process of removing clinker if such doors were provided would keep the boiler out of service for only fifteen to twenty minutes during each cleaning at intervals of twelve to eighteen hours. The doors during these tests were too high above the grate and it was very difficult to remove any clinker through them. The side door, although not so high above the grate as the front fire doors, was unhandy on account of the small space available for handling the fire tools between the boilers.

CONCLUSIONS.

The combination of boiler and furnace setting described gives good results with the North Dakota lignite. Steam can be made with a fuel efficiency of 55 to 58 per cent of the heat in the coal, and no difficulty is experienced in obtaining the full capacity of the boiler.

These results compare very favorably with the results obtained in the average plant using a good grade of bituminous coal when the heat available to the boiler is considered.

Little, if any, advantage is gained by crushing the coal by hand instead of in a power crusher.

To reduce the moisture in the coal by weathering seems to improve the economy, but these tests are not sufficient in number to determine definitely the condition of the fuel and the time required for weathering to insure the best results.

The steam blower for the ash pit is inefficient, and there is no gain in supplying superheated steam to it. A considerable saving in steam and equally good results could probably be obtained by substituting for the steam blower a fan such as is commonly used for forced draft.

PUBLICATIONS ON FUEL TESTING.

The following publications of the United States Geological Survey and the Bureau of Mines, except those to which a price is affixed, can be obtained free by applying to the Director of the Bureau of Mines, Washington, D. C. The priced publications can be purchased from the Superintendent of Documents, Government Printing Office, Washington, D. C:

UNITED STATES GEOLOGICAL SURVEY.

- BULLETIN 261. Preliminary report on the operations of the coal-testing plant of the United States Geological Survey at the Louisiana Purchase Exposition, in St. Louis, Mo., 1904; E. W. Parker, J. A. Holmes, M. R. Campbell, committee in charge. 1905. 172 pp. 10 cents.
- PROFESSIONAL PAPER 48. Report on the operations of the coal-testing plant of the United States Geological Survey at the Louisiana Purchase Exposition, St. Louis, Mo., 1904; E. W. Parker, J. A. Holmes, M. R. Campbell, committee in charge. 1906. In three parts. 1492 pp. \$1.50.
- BULLETIN 290. Preliminary report on the operations of the fuel-testing plant of the United States Geological Survey at St. Louis, Mo., 1905, by J. A. Holmes. 1906. 240 pp. 20 cents.
- BULLETIN 316. Contributions to economic geology, 1906, part 2, by M. R. Campbell and others. 1907. 543 pp. 70 cents.
- BULLETIN 323. Experimental work conducted in the chemical laboratory of the United States fuel-testing plant at St. Louis, Mo., January 1, 1905, to July 31, 1906, by N. W. Lord. 1907. 49 pp. 10 cents.
- BULLETIN 325. A study of four hundred steaming tests, made at the fuel-testing plant, St. Louis, Mo., 1904-1906, by L. P. Breckenridge. 1907. 196 pp. 10 cents.
- BULLETIN 332. Report of the United States fuel-testing plant at St. Louis, Mo., January 1, 1906, to June 30, 1907; J. A. Holmes, in charge. 1908. 299 pp. 25c.
- BULLETIN 334. The burning of coal without smoke in boiler plants; a preliminary report, by D. T. Randall. 1908. 26 pp. 5 cents. (See Bulletin 373.)
- BULLETIN 336. Washing and coking tests of coal and cupola tests of coke, by Richard Moldenke, A. W. Belden, and G. R. Delamater. 1908. 76 pp. 10 cents.
- BULLETIN 339. The purchase of coal under government and commercial specifications on the basis of its heating value, with analyses of coal delivered under government contracts, by D. T. Randall. 1908. 127 pp. 5 cents. (See Bulletin 378.)
- BULLETIN 343. Binders for coal briquets, by J. E. Mills. 1908. 56 pp.
- BULLETIN 362. Mine sampling and chemical analyses of coals tested at the United States fuel-testing plant, Norfolk, Va., in 1907, by J. S. Burrows. 1908. 23 pp. 5 cents.
- BULLETIN 363. Comparative tests of run-of-mine and briquetted coal on locomotives, including torpedo-boat tests and some foreign specifications for briquetted fuel, by W. F. M. Goes. 1908. 57 pp.
- BULLETIN 366. Tests of coal and briquets as fuel for house-heating boilers, by D. T. Randall. 1908. 44 pp.
- BULLETIN 367. Significance of drafts in steam-boiler practice, by W. T. Ray and Henry Kreisinger. 1909. 61 pp.
- BULLETIN 368. Washing and coking tests of coal at Denver, Colo., by A. W. Belden, G. R. Delamater, and J. W. Groves. 1909. 54 pp.
- BULLETIN 373. The smokeless combustion of coal in boiler plants, by T. D. Randall and H. W. Weeks. 1909. 188 pp. 20c.

- BULLETIN 378. Results of purchasing coal under government specifications, by J. S. Burrows; Burning the small sizes of anthracite for heat and power purposes, by D. T. Randall. 1909. 44 pp. 10c.
- BULLETIN 385. Briquetting tests at Norfolk, Va., by C. L. Wright. 1909. 41 pp.
- BULLETIN 392. Commercial deductions from comparisons of gasoline and alcohol tests on internal-combustion engines, by R. M. Strong. 1909. 38 pp.
- BULLETIN 393. Incidental problems in gas-producer tests, by R. H. Fernald, C. D. Smith, J. K. Clement, and H. A. Grine. 1909. 29 pp.
- BULLETIN 402. The utilization of fuel in locomotive practice, by W. F. M. Goss. 1909. 28 pp.
- BULLETIN 403. Comparative tests of run-of-mine and briquetted coal on the torpedo boat *Biddle*, by W. T. Ray and Henry Kreisinger. 1909. 49 pp.
- BULLETIN 412. Comparative tests of run-of-mine and briquetted coal on a locomotive boiler, by W. T. Ray and Henry Kreisinger. 1909. 32 pp.
- BULLETIN 416. Recent development of the producer-gas power plant in the United States, by R. H. Fernald. 1909. 82 pp.
- BULLETIN 428. The purchase of coal by the Government under specifications, with analyses of coal delivered for the fiscal year 1908-9, by George S. Pope. 1910. 82 pp.

BUREAU OF MINES.

- BULLETIN 1. The volatile matter in coal, by H. C. Porter and F. K. Ovitz. 1910. 56 pp.
- BULLETIN 2. North Dakota lignite as a fuel for power-plant boilers, by D. T. Randall and Henry Kreisinger. 1910. 42 pp.

Bulletin 3

DEPARTMENT OF THE INTERIOR

BUREAU OF MINES

JOSEPH A. HOLMES, DIRECTOR

**THE COKE INDUSTRY OF THE
UNITED STATES**

AS RELATED TO THE FOUNDRY

BY

RICHARD MOLDENKE



WASHINGTON

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1910



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THE COKE INDUSTRY OF THE UNITED STATES AS RELATED TO THE FOUNDRY.

By RICHARD MOLDENKE.

INTRODUCTION.

The investigations carried on at the fuel-testing plant of the United States Geological Survey at St. Louis in 1904-1907 included tests of the steaming and gas-producing qualities of many coals and of the possibility of their improvement by briquetting, together with washing and coking tests and tests of some of the resulting coke in the cupola. These tests were part of a general inquiry into the manner of utilizing most efficiently each of the coals under investigation.

The equipment used in the cupola tests of coke, the manner of conducting the tests, and the essential results have been described in publications of the Geological Survey,^a and only the more important conclusions to be drawn from the tests are mentioned in the following pages. The tests were made to show the fitness of the cokes for foundry purposes, but an incidental result was to show foundry men and the coke producers the advisability of studying the conditions of foundry practice. The present bulletin, published by the Bureau of Mines because the analyzing and testing of fuels have been transferred from the Geological Survey to the Bureau of Mines, briefly reviews the status of the coke industry, some features of practice in the use of coke in the foundry, the probable happenings of a cupola heat, and the reasons for modifying charging practice so that a particular coke can be used to best advantage.

COKE-MAKING INDUSTRY.

WASTES IN COKE MAKING.

Probably in no other industry, unless in the mining of coal itself, is there so much waste as in the manufacture of furnace and foundry coke. A study of all the coke-making districts of this country and a comparison of the methods used are most convincing in this respect.

^a Prof. Paper 48, pt. 3, and Bull. 336.

Particularly unfortunate is the fact that this waste is in most instances deliberate, the more economical methods being well known to those most interested. One cause for this condition is the conservatism of the trade, old methods being followed blindly; regardless of improvement elsewhere. Another cause is the necessity of utilizing the enormous investments in coke ovens, a change meaning the wiping out of millions of dollars of capital. Furthermore, the cost of the old ovens as compared with the elaborate by-product installations is comparatively low, and, finally, there is ignorance in the foundry industry in regard to cupola practice. All these causes combined have kept progress in coke making far behind as compared with the advances made in other industries.

For example, in 1907 nearly 62,000,000 tons of coal were charged into coke ovens, yielding more than 40,500,000 tons of coke. Roughly, therefore, more than 21,500,000 tons of the richest part of the coal were driven off. The probable loss involved in present coking practice has been analyzed by E. W. Parker,^a from whose statement the following figures are taken:

Coke production, 1907.

	Coal used.		Coke produced.	
	Quantity.	Value.	Quantity.	Value.
	<i>Tons.</i>		<i>Tons.</i>	
Beehive ovens	54,485,522	\$56,956,008	35,171,665	\$89,873,969
Retort ovens	7,460,587	16,874,480	6,807,879	21,666,157
	61,946,109	72,830,488	40,779,544	111,539,126

The total value of the by-products from the retort-oven coke was \$7,548,071. If it is assumed that the coal coked in beehive ovens was similar to that coked in retort ovens, and that the prices obtainable for recoverable by-products would not have been less than 80 per cent of the average prices for 1907, the value of the recoverable products wasted that year amounted to \$44,000,000, or nearly 80 per cent of the value of the coal actually used in beehive ovens.

If it is assumed further that the difference in value per ton of the beehive and the retort-oven coke was due only to difference in freight charges (the beehive ovens being mostly near the mines, the retort ovens near large cities), the total value of the 1907 output of beehive-oven coke (54,485,522 tons) if made in retort ovens close to market would have been \$135,750,000; this amount added to \$44,000,000, the assumed value of the recoverable by-products, gives nearly \$180,000,000. Deducting the difference in assumed value between

^a Coal fields of the United States, in Papers on the conservation of mineral resources: Bull. U. S. Geol. Survey No. 394, 1909, pp. 17-26.

the coal charged into retort and into beehive ovens (\$52,000,000) from the difference in value of the coke and by-products if the coal had been used in retort ovens (\$90,000,000) the remainder is \$38,000,000, which represents (minus extra expense in operating costs, up-keep, interest, etc.) the approximate net loss due to beehive-oven coke production in 1907.

The change from the old wasteful beehive oven to the modern by-product plant will not come very quickly, for it involves enormous investment as well as a systematic development of the markets to take up the tar, surplus gas, ammonia products, and benzol that will be made as the newer installations are built. We are far behind Europe in this respect; for if we should seriously undertake to bring about the use of these by-products to the extent that is logically possible, by making creosote, the aniline dyes, etc., there would be no further need to burn tar under boilers to get rid of it, and no more importations of ammonium salts, creosote, etc.

Foundry men also are in need of instruction regarding the use of retort-oven coke, which, though giving as good results as the best Connellsville of the old days, requires freedom from prejudice regarding its appearance until the operative force of a foundry is familiar with it. The excellent quality of the coke produced in some parts of the Connellsville region has probably been the greatest hindrance to advancement in coke making. When going over the best plants, one can not much blame the foundry man for wanting just that kind of coke and taking no other. The coal from which the best coke is made is, however, nearly exhausted, and in order to lengthen the life of the mines it should be mixed with poorer grades, the combination making an equally useful coke.

Another source of waste to which attention should be called is the coke breeze. At some of the retort-oven plants the breeze is systematically worked up for special industrial purposes. Thus, coke that is too small for the foundry goes to the domestic markets for household purposes; but the very fine material, corresponding to the cinders coming from a locomotive, is sold to zinc-retort operators. Very little is lost.

In the coke regions where the old methods are used, however, the coke breeze lies in immense piles, unless, indeed, it is burned as fast as made, and a considerable portion of the coal originally used goes to waste in this manner. Now, it is well known that this breeze, often brought into the market for domestic purposes and known as "crushed coke," makes the best kind of fuel for hot-air furnaces in residences; yet in the regions where it is made no one seems to think it worth while to take advantage of fuel that could be had for the mere carting. Coal evidently is still so cheap that the need of a substitute is not felt.

The coming advance in the art of briquetting fuel in this country, which will incidentally open a great market for the tar surplus complained of by some retort-oven coke interests, should also help to solve the coke-breeze problem. Undoubtedly, if this finely divided coke is studied with a view to making it more valuable metallurgically—say, by the neutralization of sulphur—it will some day become a factor of particular interest to the foundry as well as to the blast furnace.

All classes of citizens could help to encourage less wasteful methods of using coal, and our industrial experts in particular should study methods of economizing to the utmost extent, by improved processes of manufacture and use, the resources that seem likely before long to be exhausted.

PROCESSES AND METHODS OF COKE MAKING.

In order to lay a foundation for the later discussion of the proper use of coke in the foundry the processes and methods of producing coke must here be briefly described.

In general coke is made either by the beehive process or in the retort or by-product oven. In the former process all the volatile matter is lost in the atmosphere; in the latter process the volatile contents, except the gas used in heating the ovens, are recovered as surplus gas, tar, and ammonia liquor or ammonium sulphate.

In either process the quality and structure of the coke produced depends on the original composition of the coal and on the heat treatment it receives. Thus a coal high in volatile matter will naturally make a light, porous coke, the escape of the gases tending to produce such a structure. The yield of such a coal will also be much less than that of a coal higher in fixed carbon and correspondingly lower in volatile. Coal of the latter kind will make the denser, heavier cokes, which carry a burden well and are therefore more sought after for melting purposes, other factors being equal. Combination in the by-product oven of the two classes of coal, so as to give an average percentage of volatile matter, will probably get the best general results in the future, for coals of the proper composition can be brought from various sources and can be mixed and run so as always to give the same grade of coke, the mixture doing away with any particular defect the coke might have if made from any one of the coals. A good structure can then be combined in a coke with low sulphur and phosphorus for the blast furnace or with low sulphur and ash for the cupola, and a mixture giving a light, porous structure and a high yield of by-products can be used for domestic coke.

In the beehive oven the slack or ground coal (washed, if necessary, to remove excessive sulphur and slate) is dumped through the orifice

in the roof and leveled off to the desired depth. This bed may be thin if 24-hour coke is wanted, thicker if the run is to be normal, and heavy if 72-hour coke, left over Sunday, is to be obtained. Only coke of this last kind is supposed to go to the foundries. They pay for such coke, but as a matter of fact the foundry shipments are composed of coke made in twenty-four, forty-eight, and seventy-two hours, special care being taken, however, to select the best looking of each kind. Though foundry men are not hurt in any way by this substitution, as it does not matter how long the fuel was in the oven if it is of proper composition and physical structure, they should be better posted on the coking process, so that if for any reason the brand of coke they regularly use can not be had the introduction of a new one will not demoralize the plant.

The degree of heat, the time of running the oven, and the manner in which the air is admitted, of course, have a great effect on the physical structure, and where the longest time is allowed for making coke the results are apt to be best for foundry purposes. Thus, during the depression at the close of 1907 many ovens in the West Virginia coke districts were run with heavy charges, the heat lasting a whole week. The purpose was simply to keep the ovens hot and make little tonnage. Yet the coke turned out to be exceedingly hard and still of excellent structure for melting purposes. On the other hand, where ovens are hardly given time to complete the coking process the product is necessarily soft, will be high in volatile matter, and inferior even for the blast furnace.

The effect of heat treatment is even more noticeable in the retort oven. Here the coal in properly ground form is dropped into the large retorts or chambers and occasionally is stamped in, or it may be pressed into cakes and forced in from the end. The oven is tightly closed, the heat being supplied through flues in the walls of the oven. The by-products are rapidly given off, practically all of them being recovered during the first two-thirds of the process. Where recovery of these by-products is the primary object and the quality of the coke is a secondary consideration the ovens are drawn as soon as the point of greatest economy has been reached (sometimes in fifteen hours); the result is poor coke, as many blast-furnace men have found, and the result has been a prejudice against retort-oven coke. Where, however, coking is continued for the full twenty-four hours, or even for thirty hours, the coke is of very good structure, provided it has had proper attention and the coal mixture is right.

Consumers of retort-oven coke should therefore make sure that their interests are regarded where it is produced; such insistence has, in fact, so worked out that, particularly in the interest of foundries, certain by-product plants in this country are run on mixtures of coal and under heat conditions that leave nothing to be desired.

The object of this bulletin is not to treat specially of coke making, and it is assumed that the reader is familiar with the beehive as well as the retort-oven process. Another bulletin, which is in preparation by the Bureau of Mines, treats of this subject exhaustively. It is well to say here, however, that cokes of all kinds and compositions can be made by carefully mixing the coals, not all of which need be of coking varieties, and by running the ovens with skill and in accordance with a definite programme. Though considerable improvement has been made, especially in the last few years, in the mechanical part of the beehive process, the method that will ultimately survive must be one providing for by-product recovery; and the sooner the coke industry awakes to this fact the better will it be for all concerned.

USES OF COKE.

Coke is used in the foundry principally for melting iron. This process is carried on almost exclusively in the cupola, where the coke is in direct contact with the metal. It is also used for melting brass and other nonferrous metals in crucibles. Here the heat of the burning coke is imparted to the metal through the walls of the crucible and the fuel and metal are not in contact. Finally, coke has a variety of minor uses about the foundry—in fires for drying cores, heating buildings, skin-drying molds, etc. Recently coke has been introduced with great success for firing boilers within city limits, as it is smokeless, like anthracite. For the same reason it also often replaces anthracite in the annealing rooms of the malleable foundry.

COKE IN FOUNDRY MELTING.

CONSTRUCTION OF THE CUPOLA.

The apparatus used for the cupola-melting process is of the simplest construction. An upright cylinder of boiler plate is lined with fire brick to resist the high temperatures of the interior. The lower end is closed by hinged doors and made tight against molten iron by fire sand mixed with sufficient clay to hold it. A tap hole at the lowest point of the slightly sloping bottom drains out the molten metal, and perforations a little way from the bottom allow air to be driven in to promote intense combustion of the coke. The coke gives up its heat of combustion to the charges of iron, melting them down. The niceties of the process consist in the proper adjustment and shape of the orifices, called tuyeres, the pressure at which the air is forced in, the method of charging the carefully proportioned coke and metal, and the general handling of the process to get the best results.

CHARGING METHODS.

In the early times, when but little was known of the rationale of the process, it was found that when a good bed of either anthracite

coal or, later, coke was put in the bottom of the cupola and the top of this bed was a considerable height above the tuyeres, so that the first metal charged was at or above the hottest part of the fire when the blast was on, the metal would melt and could be tapped out and poured into castings. It was only necessary to keep on charging metal and fuel together into this fire, but in such proportion that the melting would proceed steadily at the rate shown by experience to be proper for the cupola.

All that later practice did was to stop the indiscriminate manner of charging and to separate metal and fuel into layers, so that the action would take place with some regularity and the mixture could be changed for different classes of work, and so that the metals could be caught with some degree of certainty during the melt. In view of the work of our early foundry men it is a question whether this change in practice has brought about a real improvement in the quality of the castings. Aside from the achievements of foundry metallurgy in the light of our present knowledge, it may be seriously questioned whether there is much truth in the general complaint of foundry men that pig iron has deteriorated in the last few decades. Pig-iron men hold that pig iron has improved and not deteriorated, and certainly modern blast-furnace practice is vastly improved, at least as to regularity in product. It would seem rather that the foundry man does not get as good results as formerly—that is, the foundry man who has no use for or does not know about the advantages of chemistry—because he can not use the pig iron as he formerly did, judging it by fracture. With the changes in furnace-casting practice the appearance of the fracture is no longer even a reasonably reliable guide.

Fortunately the introduction of chemistry, or rather metallurgy, into the foundry industry has done wonders, giving us the means to control our mixtures and start the process right, whatever we do to spoil it afterwards. Hence, also, has developed the tendency in this country to test the product not by cutting pieces out of castings, but by pouring test bars under conditions that give the metal as melted and poured into the castings the freest and best chance to show its qualities. It still remains for the foundry man to follow his craft according to his best knowledge and skill, so that the castings shall be the best that can be made with the iron so tested.

UNCERTAINTIES OF CUPOLA MELTING.

It is gradually coming to be recognized that the metal charged, be it ever so good, may be affected by a number of conditions during the melting and may turn out either as a creditable product or as a very inferior one. This is not to be wondered at. Melting depends upon the transmission of heat from the fuel to the metal at such a rate and in such degree that the solid mass of iron becomes liquid.

Iron is not one of the inert metals, such as are uninfluenced in quality by the process of melting. It would not matter whether the heat got into such a metal slowly or quickly, in one hour or five; so long as enough heat got in to do the melting the result would be the same. Iron is very much affected by the speed with which heat above the melting point is imparted to it. Furthermore, gases made intensely hot by the chemical action of burning or oxidizing carbon are also by no means inert. A chemical reaction between unconsumed oxygen and iron, silicon, carbon, manganese, etc., is certain to take place, for such reactions take place most readily at very high temperatures, and so the iron is more or less damaged.

The rate at which chemical changes proceed in the cupola, the intensity of the reactions, and the consequent need of keeping conditions just right throughout a heat are shown by the fact that under ordinary conditions it takes only one-fortieth of a second for a molecule of the oxygen in the air of the blast to pass from a tuyere through the incandescent bed and into that part of the charge too cold for chemical reactions. This brief interval is all the time the oxygen has to combine with the carbon of the coke, and (under certain conditions) with some of the manganese, silicon, or carbon of the metal charged, or with some of the iron itself.

Thus the management of foundry melting, whether in the cupola, air furnace, or open hearth, becomes a continuous struggle to get the valuable result of melting while avoiding the deleterious effect (as shown by defective castings) of oxidation of the metal.

It is no wonder, then, that there are still a few foundry men in remote parts of the world who cling to the ancient but admirable crucible process, in which fuel and metal are not in contact. This process minimizes the harmful effects of oxidation. Commercial reasons have practically eliminated the crucible process for plain cast iron, so those foundry men who want the highest grade of work select the air furnace, or hearth furnace, as it is often called, or perhaps the open-hearth furnace with regeneration of air and gas as fuel. In each of these processes the metal is in contact, not with the fuel, but only with the gases. Therefore, if the processes are carried on properly, there is a smaller chance for oxidation of the iron than in the cupola; moreover, the metal as it rolls down in little drops is not impinged upon by a current of gases containing abundant free oxygen, as it is in the cupola.

These statements do not mean that excellent metal can not be made by the cupola process—metal that will prove just as good under test as air-furnace metal of like composition—but the chances are not in favor of such success; and for the highest grade of product, especially if safety of life and limb may depend on it, such chances should really not be taken.

The few points mentioned above are enough to show that the melting process in the cupola might well be made the object of special study, and if the foundry man knows and has the proper irons and can make the required adjustments in his apparatus his study may be restricted to the coke (if that is the fuel) and the manner in which it is allowed to act on the metal to be melted.

ADJUSTMENT OF PRACTICE TO FUEL.

The writer has had a part in the efforts of associated foundry men from the beginning of what may be called the new era of the industry. He has listened to many experiences related both by veterans and by men newer to the business. All these experiences, including the foaming of enormous quantities of the slag out of the cupola doors on the charging platform and the disappearance of incredible quantities of metal during the melt, have strengthened his belief that troubles laid to the idiosyncrasy or, as some call it, the "innate cussedness" of the cupola (anyone's cupola has it when run by someone else) could really be avoided by changing the melting program to suit the fuel that is used.

Normal coke has 50 per cent cell space and has, pound for pound, practically double the volume of anthracite. Hence during the melt the downward movement of the metal within the range of the melting zone must be twice as fast when coke is used as it is with anthracite. Therefore the difference between the first of the metal to melt and the last, with respect to the influences to which they have been exposed, is greater for a bed of coke as compared with a bed of anthracite of like weight. That is, the first metal to melt is likely to be the same on either the coke bed or the bed of anthracite, but the last metal, especially if the charge is very heavy, melts much lower down in the coke-charged cupola and is necessarily subjected to greater oxidizing influences, because the blast low down in the cupola contains more free oxygen. When anthracite is used the part of the bed where the melting is actually done varies least in position.

This advantage is still retained by many foundry men in the East, where anthracite, as it is still available and no dearer than coke, may be used for the fuel bed and often mixed with coke in the subsequent charges, or even used altogether.

A study of the downward movement of the metal, as just stated, is important, as it suggests the way to make use, in regular casting practice, of practically every variety worthy of the name of foundry coke. Anthracite and coke, being practically identical in composition—that is, containing the same percentage of ash, fixed carbon, sulphur, etc.—differ radically only in physical structure. Thus, if the cell space of a coke is 50 per cent, as mentioned above, the coke has twice the volume of an equal weight of anthracite. Now, not all cokes have the same cell space, and, moreover, cokes differ in per-

centage of ash, etc. Hence a given weight of coke varies greatly in volume, and consequently the behavior of charges of iron melted with it also varies. The foundry man when changing his brand of coke makes the mistake of believing that the new kind ought to be handled in the same way as the old. It may be possible to do so if the two cokes have the same characteristics, but if they have not the foundry man is certain to get differences in results, which he usually ascribes to sulphur, high volatile, black ends, and what not. His coke dealer then hears from him. This difficulty is so marked that even when the same brand of coke is bought the year round and trouble results, investigation of the shipments often reveals substitution of coke from different companies located in the same general region. Furnaces have this difficulty to contend with more particularly.

It is therefore highly important that the foundry man get his coke from the same ovens and the same coal mines—in fact, have it as nearly the same as it can possibly be made; if a change is desirable or necessary he should be informed, so that he can adjust his work to the changed conditions.

In this respect the retort-oven coke has a great advantage. After the coals have been contracted for and the proper mixture has been established, the result will continue to be even as long as the coking process is carried out uniformly; and as the highest scientific skill is available at retort ovens, there should be little variation from the standard if the management honestly wishes none.

Another advantage of retort-oven coke is the size and shape of the lumps. A retort oven is narrow, and the coking of a charge proceeds from the walls toward the middle; the result is a mass of coke that, when quenched, breaks into short, thick pieces. The absence of thin fingers is decidedly noticeable. In comparison, the lumps of standard grades of beehive-oven coke are generally longer and thinner. Moreover, makers of retort-oven coke have carefully sorted the product before shipment, so that it has contained little breeze and the lumps have been of good shape. The size of the lumps of fuel in a cupola charge has much to do with the progress of the heat, and the absence of thin fingers accounts in a measure for the remarkable success of retort-oven coke when it was first used in foundries. Charges of coke having lumps that are uniform in shape and size give results similar to those obtained by using anthracite, which is marketed in screened sizes, because the uniform sizing of the fuel permits an even and regular penetration of the bed by the blast.

In order to adjust the process to the varying character of the coke used for melting in the cupola, it is necessary not only to change the melting conditions, for instance, by using a milder or stronger blast according as the coke is light or heavy, but also to change the relative weight of the metal and coke charges on the bed. A foundry man using heavy charges, though he may not know why, avoids coke with

a large percentage of volatile matter. Such coke is necessarily underburned and light and ignites rapidly, spreading the melting through too large a volume. Very light charges will obviate much of this trouble, as the bed of coke will be kept more nearly uniform in height, and the melting will be done at the proper point. A coke in which the volatile matter is too high is virtually a gas coke and is too troublesome for use in the foundry, at least for cupola melting. It will serve well enough for all the other purposes.

Before the question of charging is taken up it will be of interest to review the original tests made by the United States Geological Survey at St. Louis, in the exposition foundry, where a collection of cokes such as never had been got together before was tested under identical conditions of melting.

COKE TESTS BY THE GEOLOGICAL SURVEY AT ST. LOUIS.

METHODS AND RESULTS.

Foundry men will remember that several years ago an elaborate series of cupola melting tests was made at the fuel-testing plant at St. Louis on cokes made from coals sent in from all parts of the country. These cokes were made under the direction of A. W. Belden, of the United States Geological Survey, and the melting tests themselves were made under the direction of the writer, assisted by Mr. Belden as executive. By being present at nearly every test, the writer was able carefully to gage the behavior of the several cokes and to determine their merits for the foundry. The cokes were naturally selected for their probable fitness for foundry purposes; that is, those having excessive sulphur and ash were barred.

The series of tables subsequently published* gave information of direct value to the owners of the coal and demonstrated many things impossible to ascertain in ordinary foundry practice, because no individual could afford to burn up valuable iron, even if the time could be taken to do it.

In all about 190 tests were made. Three thousand pounds of metal were melted in each test. To have uniform conditions in the coke bed for each test and at the same time to suit the average coke made, the height of the bed above the tuyeres was fixed at 14 inches as the standard, and this height was kept for every test. The upper tuyeres of the cupola were not used. A melting ratio of 7 to 1 was adopted, as the diameter of the cupola was small. The coke for the bed was weighed and put in up to the proper point, as measured by a weighted wire. The weight of the several cokes for this constant bed varied from 180 to 230 pounds, showing a considerable range in specific gravity—from the very light to the heavy cokes.

* Moldenke, Richard; Belden, A. W.; and Delamater, G. R., *Washing and coking tests of coal and cupola tests of coke*: Bull. U. S. Geol. Survey No. 336, 1908, pp. 48-76.

Metal weighing four times this weight of coke was charged on this bed, to conform to the general custom of foundries, and the remaining successive charges of metal and coke were divided into four parts, so as to preserve the total ratio of 7 to 1, the coke charges varying from 50 to 62 pounds and the metal charges correspondingly.

The blast was kept at about 7 ounces, and the time when the first iron appeared at the spout was noted. This time, which was five to fifteen minutes, was interesting as indicating how the coke behaved in burning and in allowing the iron to come into the melting zone. With the best results the iron came in seven to ten minutes.

Necessarily for the extremely light and very heavy cokes such melting practice would mean disaster, and in fact the melting loss did point very markedly to such a result. Possibly this may also convince many foundry men who still think it impossible to burn up iron in the cupola. The results showed the melting losses to be from 3.2 to 52.5 per cent of the metal charged. The cupola was continually slagged off, but in the worst case so much slag was made that it flooded the tuyeres and effectually stopped operations. When the bottom was dropped no metal was found remaining. It was evident that the lighter varieties of coke burned away so rapidly that the metal came to the lower portion of the melting zone much faster than it should have come. Moreover, as the original bed went down unduly for the melting of the first charge the subsequent coke charges did not bring the bed up to proper level again, and hence a Bessemerizing of the metal took place almost from the very beginning.

In tests of the very heavy cokes it was necessary to wait a good while before the bed had burned low enough to begin melting. Necessarily at the fixed ratio of 4 to 1 the first charge of metal on the heavy coke bed was very heavy. In this case also there was an undue lowering of the bed by the heavy metal charge after the surplus coke had burned off to begin melting. What was worse, however, was that the heavy coke bed took so large a part of the total amount to be used that the upper coke charges were necessarily very small, and the practical result was not only burned metal, but an insufficient amount of coke toward the end to continue the heat, so that the bottom had to be dropped. The general result of these heats with very heavy coke was burned iron for the beginning and unmelted iron in the cupola drop.

It may be seen, therefore, that when the melting process with a normal coke gives good results it does not necessarily follow that every other coke will do likewise; yet this is what the foundry man practically assumes when he changes his coke without making trial heats to see how to use it.

VALUE OF MELTING TESTS.

Perhaps the custom that is just beginning to be introduced into this country by coke companies of sending out experts to teach

foundry men to use their particular brands of coke may obviate many of these difficulties. The retort-oven coke people were simply forced to adopt this plan, the appearance of their coke being so bad, as compared with the magnificent Connellsville product, that it took great argument to sell it at all. It should be understood that the clean, silvery appearance of beehive-oven coke as compared with the dull black cast of the retort-oven product results principally from the manner of quenching. In the beehive the quenching is done within the oven, and as little water as possible is used. In the retort oven the red-hot charge is forced out and deluged with streams of water as it drops into the receptacles for the coke. Consequently the retort-oven coke contains much more water than the beehive product. The formation of steam in the cells and the condensation and consequent drawing in of the water seem to hold it tight within, and when once water is thus included it remains. Coke exposed to rain will drain off the water to a great extent, but the occluded water of quenching stays. Sellers of retort-oven coke are therefore rather sensitive on the subject of "moisture" in the specifications, though it is hard to understand why they should expect coke prices for water. In practice, coke with a little moisture does no great harm in the cupola, although the bed coke must be dry. There is, of course, a thermal loss in driving moisture off, just as there is a great saving in the blast furnace in the use of artificially dried instead of natural air. Specifications for coke should therefore fix a maximum limit for moisture, with a penalty for any above it, to insure reasonable care in the oven practice, but this upper limit should be such that good retort-oven coke can readily keep within it.

Coke producers should therefore be urged to make thorough melting tests with their coke, so that when selling it they can advise foundry men how to get the best results with it. This plan would save enormous trouble, as there are about 6,500 foundries using coke in the United States and Canada, and but a very small percentage know how to make comparative tests of coke and draw conclusions therefrom that will guide them in its proper use. There are, on the other hand, comparatively few coke producers, or selling agents for coke producers, and it would pay these firms well to have a central testing station, or a sort of "trouble bureau," for coke tests.

BURNED IRON IN THE CUPOLA.

CAUSES OF TROUBLE.

The burning of iron in the cupola process, and in fact every melting process to a greater or less extent, is now becoming better understood. In years gone by no one could be made to believe that such a thing was possible. But anyone having to do with making the iron-silicates can realize how very little silica can carry great quanti-

ties of iron to make a thin black slag. The process can be watched very nicely in heating and forging shops, where billets of iron and steel are made ready for the hammers. The modern regenerative heating furnace keeps things at an extremely high temperature, and hence a billet once up to heat must be removed and worked quickly, as otherwise it will "waste away" very rapidly. In fact, if the furnace is too hot the outside of the billet will often be ready when the inside is still too cold, so that wasting from the surface takes place and can not then be avoided.

The surface of the billet becomes oxidized or scales heavily. The scale drops off, and on touching the white-hot sand of the bottom unites with it as a silicate of iron in the form of a thin slag that runs off. Careful operators try to keep this wasting action down to a minimum, and they collect the slag in pots, selling it to blast furnaces. It contains over 60 per cent of iron and is therefore rich, though it requires much limestone for treatment if large quantities are used.

In the bottom of the open-hearth furnace pools of iron often remain after the heat is drained out. These pools oxidize rapidly, burning away with a fine display of sparks, the oxidized material uniting with the sand bottom and making a temporarily dark spot. This action is observed especially in the malleable process with the open-hearth furnace. In the Bessemer process the burning of iron is so obvious that one has only to stand near the blow to be spattered with pellets rich in iron. In making pig iron one has only to note that the same size of furnace that makes a given quantity of good, "honest" pig iron for the foundry man is made to yield very much more pig iron when forced. Iron made thus must be oxidized slightly and produces castings weaker than they ought to be. Even in the cupola the scintillations can be seen as the drops of metal find their way to the bottom. The exact degree to which the oxidation thus brought about affects the product depends on the rapidity of the stream downward, the condition of the blast as regards unconsumed oxygen at the point where the iron is melted, and the consequent condition of the metal itself. If the metal is oxidized even slightly, its melting point is raised, and it will drop more sluggishly and receive more damage from the blast farther down. Hence the importance of nicely balancing melting conditions by proper attention to the coke and charging.

EFFECT OF BURNED METAL ON CASTINGS.

In the coke-melting tests at St. Louis another matter carefully watched was the quality of the metal cast. In every test that showed high melting loss the castings were observed to be full of pin holes near the surface, and in some tests the metal was like a big sponge. Though the silicon necessarily ran low with repeated remelting, enough remained to keep the body of the metal gray; but whenever the iron was burned the fins on all the castings were white.

Long experience in the foundry business has taught the writer that whatever cause may be put forward for such defects in castings as pin holes, excessive shrinkage, draws, and cracks, and for others due to the metal and not to the molding, practically all of them can be traced back to burned iron. Whenever steps have been taken to correct the cupola practice, these troubles have disappeared. Pin holes are due, not to the inability of the air to get out of the molds fast enough, but to the yielding up of dissolved gases in the iron at the moment of set. The first thing that happens in the mold, if the iron poured is not much overheated, is the setting of a thin skin of metal against the sand, and hence these gases in going upward and outward to escape strike the thin shell that has set and stay there. When the casting is put on the planer and the skin is removed the pin holes make their appearance and the casting is condemned. Such an outcome is particularly annoying with respect to fine varieties of rolls, on which an unblemished surface is absolutely necessary. When the metal is badly burned, as, for instance, in low-silicon heats for malleable castings, the formation of gas is so strong that the casting is honeycombed, not with pin holes as in the high-silicon iron, but with blowholes half an inch across, and the interior surface is black from oxidation.

As a further difficulty, if the metal is oxidized even very slightly its melting point is raised considerably, so that it readily freezes or, as the foundry man has it, loses its "life," and the ladles "skull." As a consequence, in pouring the gates freeze up and the castings are short poured. If the casting has been poured, the molten metal may be cut off so quickly that interior shrinkages are more than likely to occur, as no feeding can take place.

Finally, burned metal is very weak. It seems as if a coating of oxide gets between the several crystals, so that their tenacious grip on one another is lessened. Hence casting strains find the metal unprepared to resist and a crack is the result.

A little oxygen can do an immense amount of damage. The very worst burned iron ever seen by the writer contained only a few hundredths of 1 per cent. The evils of high sulphur are as nothing compared to it. Hence only a very small amount of aluminum or titanium is necessary to make the correction, more than 0.1 per cent seldom being required for addition to the ladle to produce the desired deoxidation.

All the facts outlined above show how important the subject is and why the ferro alloys are becoming such a factor in casting practice. However, prevention of this burning or oxidation is better than its correction afterwards by means of alloys of iron with titanium, aluminum, manganese, and silicon.

CUPOLA CHARGING AND ITS RELATION TO QUALITY OF METAL.**PROCESSES NOT UNDERSTOOD.**

Unfortunately, we are still in the dark regarding the actual processes in the cupola. Until means are provided to take accurately temperatures as well as samples of the gases formed at every point within the active part of the working end of the cupola we shall have only theories. It is to be hoped that some day an experimental cupola will be erected and specially arranged for such tests, so that deductions based upon determined facts may be made. In the absence of the necessary data only conjecture is possible; nevertheless, the processes described in the following paragraphs seem actually to take place.

DEPTH OF FUEL BED.

All melting occurs above the tuyeres, for the blast can find an exit only up the stack. The coke below the tuyeres is therefore only a filling and serves no other purpose, as it is not even so hot as the molten iron that collects around it before being tapped. The tuyeres are placed high or low, as the exigencies of the shop may require. Where a continual stream of very hot metal is wanted, as, for instance, in light casting work or in the molding of stove plate, the tuyeres are placed about 6 inches from the bottom and the coke filling is least in volume and height. For melting pig iron before it is Bessemerized for steel the tuyeres may be a few feet above the bottom, because a rather large tonnage of metal has to be kept between blows. In European practice the forehearth does the work of a deep crucible in the cupola, serving as a big ladle or catch-basin for the molten metal running unchecked out of the cupola spout. The forehearth, however, is intended to be heated as part of the blast arrangement; but unless the regenerative system is used it is doubtful whether high enough temperatures are obtained to do this heating to any considerable extent. The proper method would be to use a mixing furnace, such as may be seen in steel practice. This is merely a tilting open-hearth furnace, in which the metal is kept continually up to the proper temperature.

Between the least and greatest depth of the bed below the tuyeres there are of course depths adapted to various classes of work, some foundries preferring a height of 20 inches between the bottom and the tuyeres and others 14 inches, 10 inches, or whatever height gives them the space they want for the storage of metal. The quantity of coke thus used, especially in small heats, naturally affects the melting ratio; if this fact is forgotten and a high ratio is arbitrarily selected, the heavy bed often leaves too little coke for the upper charges, as, for example, in the tests at St. Louis with heavy cokes. In long heats this factor is not so apparent.

SMALL CHARGES.

The portion of the bed above the tuyeres is next to be considered. Here the actual work of melting is done. The cold air entering the cupola at once attacks the coke and delivers up its oxygen, which, by the time it is all consumed, has traversed a certain distance inward and upward. This distance probably limits the melting zone, the shape of which no one can tell until experiments such as were referred to above have been made. The effect of this zone is seen along the lining, and the more defined and the narrower it is the better are the results. If the zone, as shown by the cutting action on the lining, spreads over a considerable height, the melting is irregular. The lining should be fairly clean throughout, easily chipped out, and quickly daubed up. Otherwise practice has been defective.

This melting zone may be in the shape of a parabola flaring upward along the sides or, if Mr. West's center blast is used so that the blast pressure is more nearly equally distributed, its upper limit may approach a flat plane; in either case the highest temperature is reached at a point somewhat above the entry of the air, and above that point the heat diminishes. It is in the melting zone that the reaction between the oxygen of the blast and the carbon of the coke bed is most intense, and it is here, probably, that the formation of carbon dioxide is greatest. Here, too, the cupola temperatures reach a maximum, probably about 3,500° F. Below the narrow limits of the melting zone the temperatures are less because the combustion of the carbon, its reaction with oxygen to form carbon dioxide, is incomplete. Above the melting zone the temperature falls because, if the coke bed is thick enough, the carbon dioxide formed in the melting zone has to pass through incandescent fuel, and the glowing carbon reacts with the carbon dioxide to form carbon monoxide, the gas that may be seen burning in the upper part of the cupola. This reaction takes up heat, and hence within a distance of 3 feet, if the coke bed is so thick, the temperature falls to about 2,100° F.

It is evident that if the thickness of the coke bed is so proportioned that all melting takes place within the melting zone, the melting will be done quickly and well, for the melting point of the iron ranges between 1,900° and 2,400° F. If, however, the coke bed is excessively high, some iron will begin to melt as soon as the coke bed has burned away enough to bring the temperature above the melting point, and will melt slowly. As a result, cold metal will drop into the crucible of the cupola, causing the many annoyances of cold metal in pouring. The best economy is naturally had if the metal is just at that point of highest temperature, and not much below or above it. As the heat progresses the coke is rapidly burned away. Hence the bed is lowered and the metal sinks; the next charge of coke is intended to restore the bed to its original height so that each successive charge of metal may

melt in the same position in the cupola. Under the best conditions the lower part of the metal charge melts in the upper part of the melting zone, and the upper part of the metal charge melts in the lower part of the melting zone. Hence if a metal charge is too heavy some of the iron is melted below the normal limits of the melting zone, and if a charge is unnecessarily light all the iron is melted in the upper part of the melting zone. If each charge melted at the same level there would be no more to say on the subject; and the fact that it did happen in former times, when no attempt was made to charge in layers, shows that perhaps we have not progressed so much as we suppose.

If, as there is reason to believe, the melting is best done at the point of highest temperature, the point at which the last portion of the usual heavy first metal charge must melt, owing to the sinking of the bed and the metal, is considerably too low in the cupola. The heat may be high enough there, but the gases in the cupola near the tuyeres are not yet free from uncombined oxygen. As a result the metal must suffer somewhat by contact with this free oxygen at so high a temperature. If this statement of conditions is in any degree true, the remedy is to make the charges as small as possible, so that the fluctuation in position of the effective part of the coke bed may be as slight as possible, and the cutting action in the cupola limited to a height of a few inches instead of feet.

UNIFORM CHARGES.

Inasmuch as only the very top of the coke bed is effective for proper nonoxidizing melting if it was made right at first, the question arises why it is the prevailing practice to make the first lot of metal several times as heavy as those succeeding. The obvious reason is plausible enough at first glance—namely, that a lot of metal ought to go on top of a lot of coke in the bed. It is forgotten how little of that big bed of coke really does the work of melting.

The damage caused by the big first charge of iron may best be shown by figures. Let us assume that every 1,000 pounds of iron in melting uses up 1 inch of the coke bed. If the first big charge of metal is 6,000 pounds, 6 inches of the coke will have been burned away when the metal is melted. If iron is really damaged by oxidation, the last 3,000 pounds of that charge is not quite so good as the first 3,000. Now, the first intermediate charge of coke comes down. Does that restore the 6 inches of the bed? By no means. It is merely enough to melt the next and much smaller charge of metal. Let us say it restores the bed 3 inches and disposes of a succeeding charge of 3,000 pounds of metal. The bed, restored only halfway up to its original position in the melting zone, burns down 3 inches again in melting the second charge, nearly all of that iron being melted at a point where it is affected by unconsumed oxygen in the blast. This

restoration and melting is repeated until the end of the heat, and the practical effect of the big first charge is to give an inferior grade of iron for the entire heat, unless extra coke between the charges brings the bed up to proper level.

Many foundry men have recognized this point, and now run their cupolas with charges of metal alike from first to last; and the writer has persistently advocated this method as the first step in solving the problem of using all kinds of foundry coke successfully.

But if there is any advantage in keeping the fluctuations of the bed uniform, there is additional benefit in making this fluctuation as small as possible; hence the writer's second recommendation is to make the charges so small that the layers of coke are only thick enough to cover the layers of metal, unless definite knowledge regarding the range of the melting zone of a cupola permits heavier charges.

ADJUSTMENT OF BED.

Another very important point remains for discussion. The bed must be of the right height to start the melting properly; otherwise all subsequent care is useless. Many foundry men have adopted the method outlined above of melting by small charges, most of them successfully from the start, but others have not succeeded until attention was called to the height of their bed. Practically the only way to know when this height is right is to observe the time between "blast on" and "first iron." Every foundry man knows that when this time is about seven to ten, preferably eight, minutes he gets his best results. So in determining the right height of bed, if iron comes at a time within these limits it is safe to continue to charge as in that heat. If the iron comes, say, in fifteen minutes, the bed is too thick; if it comes in four or five minutes, the bed is too thin. Thus this matter can be attended to at first. In a new cupola the bed can purposely be made too thick in the first heat and reduced subsequently until the proper time of "first iron" is reached. This "first iron" means the first metal running continuously just previous to "stopping up," and not the few random drops that often come in five minutes. Where in practice the tap hole is closed when the blast is put on, it had better be left open, for a few days, until the action of the cupola in this respect is known.

In making the estimate of the correct time from "blast on" to "first iron," it was assumed that the charging was done on a fuel bed well burned through, with all the wood consumed. The writer has noted instances in which about a third of the bed coke was put on and burned through. Then the remaining two-thirds was added, the cupola charged, and the blast turned on. In such instances the first few minutes of the observed time to "first iron" were taken up by the fuel bed burning through and did not form any part of the melting

time. Iron in eight minutes was really iron in four or five minutes, consequently the results were disappointing.

Too heavy a bed has nearly as bad an effect as one too thin. The thin bed, as has already been explained, has just the same effect as a very heavy first charge. The entire process of melting throughout the heat takes place too low down and in a more or less oxidizing atmosphere. If the intermediate charges of coke used were not as a rule slightly heavier than necessary, thus causing the fuel bed gradually to rise, a low bed together with a heavy first charge of metal would give serious trouble throughout the heat. As it is, these conditions may account for much of the badness of the "first iron," commonly laid altogether to sulphur.

On the other hand, too thick a bed unduly delays the metal. The bed has to burn away, and while it burns some of the stock melts slowly too high in the cupola and hence is exposed too long to the influence of the cupola gases. The melting is therefore slow, and the iron may be slightly burned and is apt to be cold. Slow iron nearly always means inferior iron. The evils of too heavy initial charges are perhaps most noticeable in air-furnace and open-hearth practice, where a slow heat is an abomination.

If the bed is too thick, a heavy first charge of metal keeps the succeeding charges of iron from melting too high in the cupola and reduces the danger from burned iron. The writer, indeed, has observed a number of instances of excellent results obtained, with a thick bed and heavy first charge, with very small intermediate charges. Such practice does not seem good, however, as some of the "first iron" will be dull. Better have a proper bed and uniform charges.

In changing from the current practice of high fuel bed and heavy initial charge of metal, it will be noticed that a little while after the start the melting always becomes slower; that is, of course, if the same ratio of coke to iron is used for the intermediate charges in the two methods. This is because some of the coke which was used up under the old conditions to make slag and cut the lining becomes available for melting, less fuel is needed, and the fuel bed rises. This thickening of the fuel bed and slow melting is easily remedied, when observed, without doing harm anywhere and with a saving of coke by changing the melting ratio; that is, cutting down the intermediate coke charges slightly.

It is but natural that the system of cupola charging above outlined, which has been before the foundry public for several years, should have been tried out by many foundry men with varying success. Among the hundreds of instances of trials in large and in small cupolas that have come to the attention of the writer, whenever failures or indifferent results were reported, investigation always indicated something wrong with the bed charge, or a lack of care in the charging itself. Melters object to the extra labor of weighing out small charges,

preferring their comfort to clean, sound iron. In some instances, indeed, melters have deliberately added extra coke in order to show that melting became slower by the small-charge method. However, no matter what the cupola or what the class of castings made, a reduction in the size of the charges always works to advantage, and wherever slower melting has been noticed a slight reduction in the intermediate coke charges has always restored the proper rate of melting. Naturally, it takes more care to get an even distribution of small than of large charges, but the reduction in scrap castings is surely worth some effort.

Again, it does not follow that burned iron always results from very heavy charges. When the conditions become such that the molten iron is attacked by oxygen, the iron may be protected by the manganese in the charge. Silicon gives similar but less protection. The foundry man merely has to remember that when the metal is slightly too low in the melting zone of the cupola the same conditions obtain, though in far smaller degree, as in the Bessemer process, in which manganese, silicon, and carbon, and with them more or less iron, are burned out. What the foundry man should look to is that his cupola does not become a sort of converter during any part of the melt.

In conclusion, it may be said that the logical method of cupola charging after the bed is in would be to do away altogether with separate metal and coke charges, and yet keep the proper ratio of iron to coke. This ratio can be maintained by weighing out metal and coke in separate piles on the platform and charging a small quantity at a time from each pile, after the manner of proportioning sand and facing materials. The claim has been made that this method, the one used by our foundry-men forefathers, means slower melting, but careful trial has shown that such is not the case. However, the discomfort and cost of the method will preclude its use.

The writer has advanced another suggestion, that to reduce the cost of charging a cupola to a minimum the charges should be made up in the yard. They should be carefully laid in special charging buckets with swinging bottom doors to drop the charges exactly as laid. If the cupola is cut off at the platform, one man in the yard with a reversible hoisting engine and a revolving jib crane can handle the charges. This plan has been tried at a large foundry with excellent results. A suitable hood and stack above the cut-off cupola will carry off escaping gases.

SUMMARY.

The probable processes of a cupola heat have now been considered, and the lessons derivable therefrom if the assumptions are correctly made. In this description of a cupola heat the foundry man may find suggested a method by which he can make use of practically any kind of coke. If the coke is heavy—almost like anthracite—he can

use fairly heavy charges, as the fluctuations of the melting zone will be slight. If the available coke is light, he absolutely must take small charges, the smaller the better, for large charges would result in too strong combustion and perhaps in red-hot coke near the charging doors. The limit for lightness is set only by coke so light that combustion takes place too rapidly and the iron can not take up the heat fast enough. This limit is quickly found in practice. The methods of making all varieties of coke, and particularly those made in retort ovens, are now fairly well settled, and hence foundry coke is made a specialty at only those plants that can obtain the right kinds of coal for the purpose; that is, these coke makers have coals of various kinds available, which when ground and mixed give the desired result.

As retort-oven coke, it is to be hoped, will be the foundry fuel of the future, the coke makers, by careful study of details, will probably bring about a very uniform product, with so slight variations in composition and physical structure that a method of operation in the cupola such as that outlined above will give uniform satisfaction to the user. In the meantime, however, we have to do with existing conditions. We have coke with low and with high sulphur, coke with low and with high ash, and occasionally coke so soft as to be very near the danger limit. By the very nature of the process of making it coke can not be uniform in structure, and only careful selection from the ovens as they are discharged gives the foundry man what he is paying for. In brisk times this selection is likely to be made less conscientiously, coke forks being discarded or their prongs made closer. Many coke tests have shown conclusively that much can be done to improve a coke by adapting the process of making it to the requirements. This matter will be treated more fully in another bulletin of the Bureau of Mines.

COKE DISTRICTS OF THE UNITED STATES

GENERAL STATEMENT.

Fortunately for the foundry industry it is possible in practically all the important centers of the industry to get good cokes at reasonable prices through competitive rates from the several coal fields. The foundry man, however, who is so placed that he can get, for example, a Clinch Valley coke cheaply, but insists upon having Connellsville coke at a dollar or two higher, incurs a direct and avoidable loss.

Coal-washing methods have now progressed so far that it is possible to make very creditable foundry coke out of what was formerly considered almost too poor material for the blast furnace. Hence, if the producers give proper attention to the wants of the foundry and the users of coke take into account the differences in its structure and composition, with existing facilities for shipment, there should

be little trouble in the marketing of coke from any part of the country. It will be well, therefore, to describe briefly the coking districts of the country and point out some of the characteristics of the coals to be found in each.

Coal from five of the seven great fields of the country is used for the manufacture of coke. These fields are the Appalachian field, embracing Pennsylvania, Virginia, West Virginia, Ohio, Tennessee, Georgia, Alabama, and eastern Kentucky; the eastern interior field, in Illinois, Indiana, and western Kentucky; the western interior field, in Iowa, Kansas, Missouri, Nebraska, Arkansas, Oklahoma, and Texas; the Rocky Mountain field, in Colorado, Montana, Wyoming, Utah, and New Mexico; and the Pacific coast field, in Washington.

DESCRIPTION BY STATES.

ALABAMA.

Alabama is one of the large producers of coke and has an advantage in home markets. Its coal is rather high in impurities, and nearly all the slack and more than half the run-of-mine coal used for coking is previously washed. Probably the chief cause of objection to Alabama coke is the rather high sulphur content, which is injurious for stove castings and similar articles. Otherwise the coke of Alabama is used satisfactorily for the foundry. Alabama coke has about the following composition:

Average composition of Alabama coke.

	From run-of-mine coal.	From washed slack.
Moisture.....	1.34	0.75
Volatile matter.....	1.08	.75
Fixed carbon.....	83.85	86.00
Ash.....	14.28	11.50
Sulphur.....	1.80	.90

The analyses show up better for coke made from washed coal.

COLORADO.

Practically all coal from Colorado used for coke purposes is washed. Average analysis is about as follows:

Average analysis of Colorado coke.

Moisture.....	0.44
Volatile matter.....	1.31
Fixed carbon.....	82.18
Ash.....	16.07
Sulphur.....	.44

The coke should be improved with respect to its high ash by better development of the washery practice.

GEORGIA.

Very little coke is made in Georgia, but that little is good. The industry is confined to the extreme northwestern corner, in Dade County; "Durham" coke is known, in the market which it reaches, as a good low-sulphur foundry coke, easily operated.

ILLINOIS.

In Illinois much foundry coke is made in by-product ovens from coals drawn from West Virginia. This coke has become standard for foundry practice in northern Illinois and tributary regions. The Illinois coal itself gives a rather poor coke even when washed, though doubtless it can be used to advantage by mixing with other coal possessing better coking quality. An analysis of a coke made from a washed Illinois coal follows:

Analysis of a coke made from a washed Illinois coal.

Moisture	2.78
Volatile matter.....	.74
Fixed carbon.....	83.35
Ash	13.13
Sulphur	2.49

In spite of its quality this coke has its uses, though probably one would do well to keep clear of it for ordinary foundry work. Foundry men will recognize in the above analysis a material much like that which they sometimes get during coke famines.

KENTUCKY.

Kentucky draws its supplies of coal from two of the great coal fields. Most of the coke is made in the western part. The analysis of Kentucky coke shows normal components except the sulphur, which runs above 1 per cent and sometimes nearly to 2 per cent. The sulphur in the coal is chiefly in the form of pyrite, much of which is eliminated by washing.

NEW MEXICO.

New Mexico is becoming an important factor in the coke production of the West, as one sees on visiting its coal regions. The coal is so dirty, however, that for coking purposes it must be washed, and when it is so treated some analyses still show over 10 per cent of ash. The sulphur content is rather low, being between 0.60 and 0.70 per cent.

The great coke plant at Dawson, N. Mex., is interesting. The gases from the modified beehive ovens are used for raising steam for the plant, but the other by-products are lost.

OHIO.

Ohio is coming up as a coke-producing State, though not so rapidly as it should, probably on account of the proximity of the Pennsylvania fields. Many of the coals have to be washed, and the sulphur and ash are generally a little high.

PENNSYLVANIA.

Pennsylvania is, of course, the banner State for coke. Coke is made in ten districts that are geographically distinct. The amount of slack that is washed before coking is considerable, but not so large as in other coal fields. Nearly all of the coal mined in the Connellsville district is used for coke making, and most of the coal so used is unwashed run-of-mine. As detailed statements of the statistics can be found in the volumes of Mineral Resources annually issued by the United States Geological Survey, it will suffice here to give the range in composition.

Range of composition of Pennsylvania cokes.

Moisture	0.23 to 0.91
Volatile matter29 to 2.26
Fixed carbon	92.53 to 80.84
Ash	6.95 to 15.99
Sulphur81 to 1.87

The upper limits for ash, sulphur, and volatile matter denote nearly extreme cases either of imperfectly made coke or of coke made from coal that is not generally used for the purpose. As the foundry man is liable to have such coke sent him, it is included in the statement.

TENNESSEE.

The bulk of the coal used to make coke in Tennessee is washed. In fact, all the slack is so treated before coking. Washing is necessary on account of the bone and the occasionally high sulphur. The coke analyses, which reflect these properties, are as follows:

Range of composition of Tennessee cokes.

Moisture	0.22 to 1.67
Volatile matter11 to 1.60
Fixed carbon	92.44 to 76.87
Ash	7.23 to 19.86
Sulphur61 to 2.45

This statement shows plainly the necessity for washing, but also the fact that very good coke is to be had.

VIRGINIA.

The southwestern portion of Virginia is rapidly becoming an important coke center. The coals are high grade, producing a coke comparable with those from the Flat Top and New River districts of West Virginia. The range of the following analyses indicates what excellent material the State produces:

Range of composition of Virginia cokes.

Moisture	0.16 to 1.52
Volatile matter80 to 1.67
Fixed carbon	93.24 to 88.52
Ash	5.80 to 8.29
Sulphur42 to 1.02

WASHINGTON.

The coke industry of Washington, though not large, is important, not so much for its quality as for the fact that metallurgical coke is made at all on the Pacific coast. The coal for coke making is all washed. The importance of this treatment is shown by the following analysis of a Washington coke the coal for which had not been washed.

Composition of a Washington coke from unwashed coal.

Moisture	1. 02
Volatile matter	2. 10
Fixed carbon	77. 53
Ash	19. 35
Sulphur 44

Everything in this coke will pass except the ash and the volatile matter, the first of which can be reduced by washing and the second by suitable changes in the coking process.

WEST VIRGINIA.

West Virginia is the second largest producer of coke in the country. The quality of the coal of this State is shown by the fact that the greater part of its coke is made from slack, but little of which has to be washed. Hence the following range of analyses is interesting:

Range of composition of West Virginia cokes.

Moisture	0. 07 to 0. 60
Volatile matter 46 to 2. 35
Fixed carbon	95. 47 to 84. 09
Ash	4. 00 to 12. 96
Sulphur 53 to 2. 28

VALUE OF A STANDARD COMPOSITION.

It may be useful to give a desirable composition for foundry coke, so that a foundry man can compare it with the analyses given above for the several States and with the coke that he purchases. This function really should be performed by a standard specification, and the fixing of such a standard, it is hoped, will some day be carried out in a manner acceptable to all interests concerned. The following composition, however, would be considered excellent—better, in fact, than is actually required:

Desirable composition for foundry coke.

Moisture	0. 50
Volatile matter 75
Fixed carbon	89. 75
Ash	9. 00
Sulphur 70

PUBLICATIONS ON FUEL TESTING.

The following publications, except those to which a price is affixed, can be obtained free by applying to the Director of the Bureau of Mines, Washington, D. C. The priced publications can be purchased from the Superintendent of Documents, Government Printing Office, Washington, D. C.:

PUBLICATIONS OF THE UNITED STATES GEOLOGICAL SURVEY.

BULLETIN 261. Preliminary report on the operations of the coal-testing plant of the United States Geological Survey at the Louisiana Purchase Exposition, in St. Louis, Mo., 1904; E. W. Parker, J. A. Holmes, M. R. Campbell, committee in charge. 1905. 172 pp. 10 cents.

PROFESSIONAL PAPER 48. Report on the operations of the coal-testing plant of the United States Geological Survey at the Louisiana Purchase Exposition, St. Louis, Mo., 1904; E. W. Parker, J. A. Holmes, M. R. Campbell, committee in charge. 1906. In three parts. 1492 pp., 13 pls. \$1.50.

BULLETIN 290. Preliminary report on the operations of the fuel-testing plant of the United States Geological Survey at St. Louis, Mo., 1905, by J. A. Holmes. 1906. 240 pp. 20 cents.

BULLETIN 323. Experimental work conducted in the chemical laboratory of the United States fuel-testing plant at St. Louis, Mo., January 1, 1905, to July 31, 1906, by N. W. Lord. 1907. 49 pp. 10 cents.

BULLETIN 325. A study of four hundred steaming tests made at the fuel-testing plant, St. Louis, Mo., 1904, 1905, and 1906, by L. P. Breckenridge. 1907. 196 pp. 20 cents.

BULLETIN 332. Report of the United States fuel-testing plant at St. Louis, Mo., January 1, 1906, to June 30, 1907; J. A. Holmes, in charge. 1908. 299 pp. 25 cents.

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BULLETIN 403. Comparative tests of run-of-mine and briquetted coal on the torpedo boat *Biddle*, by Walter T. Ray and Henry Kreislinger. 1909. 49 pp.

BULLETIN 412. Tests of run-of-mine and briquetted coal in a locomotive boiler, by Walter T. Ray and Henry Kreislinger. 1909. 32 pp.

BULLETIN 416. Recent development of the producer-gas power plant in the United States, by R. H. Fernald. 1909. 82 pp., 2 pls. 15 cents.

BULLETIN 428. The purchase of coal by the Government under specifications, with analyses of coal delivered for the fiscal year 1908-9, by G. S. Pope. 1910. 80 pp. 10 cents.

PUBLICATIONS OF THE BUREAU OF MINES.

BULLETIN 1. The volatile matter of coal, by H. C. Porter and F. K. Ovitz. 1910. 56 pp. 1 pl.

BULLETIN 2. North Dakota lignite as a fuel for power-plant boilers, by D. T. Randall and Henry Kreislinger. 1910. 42 pp. 1 pl.

Bulletin 4

DEPARTMENT OF THE INTERIOR

BUREAU OF MINES

JOSEPH A. HOLMES, DIRECTOR

**FEATURES OF
PRODUCER-GAS POWER-PLANT
DEVELOPMENT IN EUROPE**

BY

R. H. FERNALD



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FEATURES OF PRODUCER-GAS POWER-PLANT DEVELOPMENT IN EUROPE.

By ROBERT HEYWOOD FERNALD.

INTRODUCTION.

The analyzing and testing of mineral fuels as carried on by the United States Geological Survey included trials of coal, lignite, and peat in a gas producer. These tests were made in connection with steaming and other tests, the object of the investigation being to find out in what ways the mineral fuels in the United States could be utilized most efficiently, and by what means deposits of low-grade fuel that were lying undeveloped or were being wasted could be made of immediate or prospective importance as assets of the nation's mineral wealth.

The tests of fuels for producer-gas manufacture began in 1904 and were continued by the United States Geological Survey up to July 1, 1910, on which date all of the fuel-testing work being carried on by the Survey was transferred by act of Congress to the Bureau of Mines. Results of some of the producer-gas tests made before that date have been given in the Geological Survey publications listed at the end of this report; results of other tests will be given in bulletins to be issued by the Bureau of Mines.

In the course of its investigations the Geological Survey collected information relating to the development of producer-gas power plants in the United States, giving particular attention to the fuels used, the efficiencies obtained, and the operating troubles or other drawbacks reported by owners. To get further particulars regarding the possibility of utilizing fuels undeveloped or wasted, the writer was authorized to visit those European countries where gas producers and gas engines are more widely used than in the United States and to inspect those plants which had been most successful in utilizing poor fuels. The writer made this trip of inspection in the fall of 1908 and visited plants in ten countries during a two-month journey.

Because the limited time available would not permit a careful inspection, much less a detailed study of all the plants visited, the writer has not aimed in this bulletin to present a comprehensive review of producer-gas power-plant development in Europe, or even to give such conclusions as might be drawn from what he saw, but has simply described some interesting features of European practice that attract the attention of even a casual observer. Inasmuch as his journey

was made with the special purpose of studying the utilization of low-grade fuels, only passing attention was given to producer plants of standard types using anthracite or other high-grade fuel. For that reason, and also for the reason that the reliability of producers running on anthracite is well established, this report merely outlines certain important features of European practice with suction producers using anthracite or coke.

The writer takes this opportunity to acknowledge the courtesies shown him at the plants he visited, and to thank most cordially the many persons who in one way or another aided him to obtain the information he sought.

USE OF ANTHRACITE OR COKE.

Suction producers fed with anthracite or coke are in general use at small power plants in Great Britain and on the Continent. They seem to give satisfaction, although their operating cost is apparently somewhat greater than the figures stated by the manufacturers. The number of producer-gas power plants in England has become so large that it is said to have caused an advance in the price of anthracite. In one city I was told that the price of a grade of anthracite had been \$4.85 a ton before the introduction of gas-producer plants, but had risen to \$7.25 since.

Coke is used abroad much more generally than in this country. Many users say that it gives best results when it is crushed to walnut size. Some English power-plant owners, to reduce fuel bills, used a mixture of the two fuels (coke and anthracite) in their producers, as the price of coke in 1908 was about \$3.75 a ton.

A manufacturer of a suction plant in general use advertises "20 h. p. hours for 1d." on coke. The writer investigated one of these plants and was informed by the owner that although coke cost him only about one-half as much as anthracite he was obliged to use twice as much, so that the expense for fuel was practically the same whichever fuel he used. Under average conditions the fuel cost of power was 1 penny for about 9 horsepower-hours. Under the best conditions the plant might develop 12 or 14 horsepower-hours on 1 penny's worth of fuel, but would not develop 20 horsepower-hours on that quantity.

In Belgium the anthracite generally used in suction plants seems to give entire satisfaction. At some plants it was reported that the anthracite contained 6 to 8 per cent ash, practically no sulphur, and little tar. The coal shows little tendency to cake, and its average heat value is 13,500 B. t. u. per pound. It costs \$3 to \$4 per ton at the mine.

In Germany gas producers of both the suction and pressure types burn anthracite, and all kinds and sizes of gas engines are built to run on producer gas. In 1908 one company was turning out 400 engines a month, most of which were to use this gas.

Small producer-gas plants of the suction type that used good anthracite were reported to require from 0.7 to 1 pound of water for the vaporizer and from 10 to 15 pounds of scrubber water for each pound of coal fired.

One practice of some of the engine manufacturers in Europe that deserves commendation is not to overrate their engines. For example, many engines rated at 50 will show 60 brake horsepower continuously under test, and those rated at 35 to 40 will show 50 horsepower. This gives an overload capacity of apparently 20 to 30 per cent.

Some of the companies that formerly made all their producer-gas engines of the hit-and-miss type now make all their large ones of the constant-mixture, throttled type.

Some manufacturers showed suction-type producer outfits especially designed for burning anthracite screenings. These outfits seemed to be of serviceable design and construction, but the author saw none in operation outside of the works of the various manufacturers.

PLANT AT SCHEVENINGEN, HOLLAND.

One of the most interesting of the suction-type plants burning anthracite which were visited is the central lighting plant at Scheveningen, Holland. The details of this are noted below:

Number of producers: 6.

Type of producers: Suction.

Rated capacity of producers: 350 horsepower each.

Number of engines: 5.

Make of engines: Nürnberg.

Type of engines: Horizontal, double-acting, tandem, direct connected to 235-kw. generators.

Rated capacity of engines: 350 horsepower each.

Engines in operation: 1 by day, 4 at night.

Use of power: Lighting city.

Service hours a day: Twenty-four hours for one engine; variable number at night for the others.

Number of men required: At night, 1 in producer room, 3 in engine room, and 1 at switchboard.

Fuel used: Selected lump anthracite; 4, 6, and 8 inch lumps.

The scrubbers contain wooden slats and a water spray.

A special tar extractor used consists of iron plates full of small holes. These plates revolve slowly and dip in a warmed hydrocarbon oil that dissolves the tar.

Large purifiers containing iron oxide and shavings are used for removing the sulphur.

An exhaustor draws the waste gases from the engines and forces them into a chimney, thus reducing the back pressure on the engines.

A particularly important feature of the plant is the setting of the engines in such manner that all parts are easily accessible.

At the time of the writer's visit in 1908 this plant had been in operation four and one-half years and had given excellent service.

USE OF BITUMINOUS COAL.

At the large producer-gas power plants in Europe bituminous coal is generally used, both at those which save by-products and at those which do not. With one or two notable exceptions, however, there seems to be no attempt to utilize low-grade fuels. In fact, the coals burned in power-gas producers are apparently especially selected for such use. Most of the manufacturers state that they confine their attention to making producers for short-flaming coals that are low in ash and in tar and do not cake, although one manufacturer claims his producers give no trouble with high-ash fuels and believes he can run them on coal containing an extremely high ash content, provided the coal is not a caking variety.

The cost of average grades of bituminous coal in one section of England is given as \$2.25 per ton and in Sweden as about \$3.75.

PLANT AT A TOWN IN WALES.

At an installation in Wales of the by-product type, which comprises two pressure-producer units of 1,250 horsepower each, the engineer stated that the coal used contained 23 per cent ash, 27 per cent volatile matter, and 2 to 3 per cent sulphur. In 1908, when one-half of this plant was in operation, none of the sulphur was removed from the gas. At this plant the producers have asbestos jackets, and the engineer claimed that these jackets increased the efficiency 5 per cent. Some of the details of this plant are as follows:

Number of producers: 2. (Only one in use at time of visit, the second having just been erected.)

Make of producers: Crossley.

Type of producers: Pressure.

Rated capacity of producers: (10 feet in diameter) 1,250 horsepower each.

Number of engines: 4.

Make of engines: Crossley.

Type of engines: Horizontal, two-cylinder, opposed, single-acting.

Rated capacity of engines: 350 horsepower each.

Load carried: 350 horsepower at the time of my visit.

Use of power: Electric power about works; gas also used for heating purposes.

Number of men required: Engineer in charge, 2 men on the producers and 1 in the engine room, each shift, when running full load.

Methods of charging producers: At full load the fuel is charged and the fuel bed poked about every hour, but at times every forty-five minutes.

Kind of fuel: Bituminous coal reported to contain 23 per cent ash and 27 per cent volatile matter.

The producers operate on the Taylor principle. Between the gas generator and the economizer is a baffle box that prevents dust and other impurities from passing into and clogging the economizer pipe. The economizer and the coke scrubbers are of the design usually made for such producers, and the tar extractor is of the centrifugal type. The tar thrown out by the extractor proves a valuable by-product, as indicated on page 12.

The process of cleaning the gas and some other features of the plant are especially interesting. The gas after passing through the tar extractor goes to a sawdust scrubber, which removes tar and dust as well as moisture. There are two of these sawdust scrubbers, either of which can be by-passed and run independently to facilitate the cleaning of the other.

No gas tank is used. A small holder, about 30 inches in diameter, is connected to the gas pipe supplying the engine and to an electric resistance switch which controls the motor for the Root blower that gives the blast for the gas generator. When the pressure in the gas main decreases the holder falls, pulls the switch, and speeds up the blower.

The necessary steam for the producer is made in an auxiliary boiler heated by the gases from the producer. It is claimed that by means of this special arrangement for generating steam in a "hot-gas" boiler through the sensible heat of the gases only a small percentage of the total steam required has to come from the independent auxiliary boiler. When the load is light no steam is furnished by the "hot-gas" boiler, but with full load this boiler supplies about 0.7 of all the steam required.

The grate in the gas generator is conical and of the revolving type. The continuous removal of ashes is made possible by a water seal. In the operation of the producer the ash bed is kept about $3\frac{1}{2}$ or 4 feet thick and the fuel bed about 14 feet thick.

One of the special advantages claimed by the manufacturers for this producer is that low-grade fuels containing high percentages of noncombustible matter can be handled successfully. Care must be taken, however, not to use a caking coal.

The following illustration (fig. 1), kindly supplied by Messrs. Crossley Brothers, Manchester, England, shows the principal features of the plant described. The foundations for the new half of the plant are shown in the foreground.

RECOVERY OF BY-PRODUCTS.

Owing to the fact that the wholesale price of sulphate of ammonia in the principal markets is from \$55 to \$60 a ton, the recovery of ammonia as a by-product in the manufacture of producer gas is a very tempting project. The possible financial return seems especially inviting when one realizes that a ton of the ordinary grades of bituminous coal yields when gasified about 90 pounds of sulphate of ammonia. In addition, it is often possible to find a market for the pitch and tar produced.

Anthracite is not suited for any profitable ammonia recovery process, owing to its small percentage of nitrogen and also to its high first cost; but the cheaper grades of bituminous coal, with an average content of about 1.3 per cent nitrogen, are especially adapted to the

successful operation of by-product plants, provided the plants are of sufficient capacity to reduce the operating expenses and fixed charges per unit of output to a reasonable figure. Attempts to recover by-products at plants of small horsepower have not proved successful financially. The impression the writer gathered abroad is that no attempt should be made to recover by-products in plants of less capacity than 4,000 horsepower, although by-product recovery plants of much smaller size are in operation.

MOND-GAS PLANT AT DUDLEY PORT, ENGLAND.

One of the most interesting by-product installations in Europe is the 16,000-horsepower Mond-gas plant at Dudley Port, South

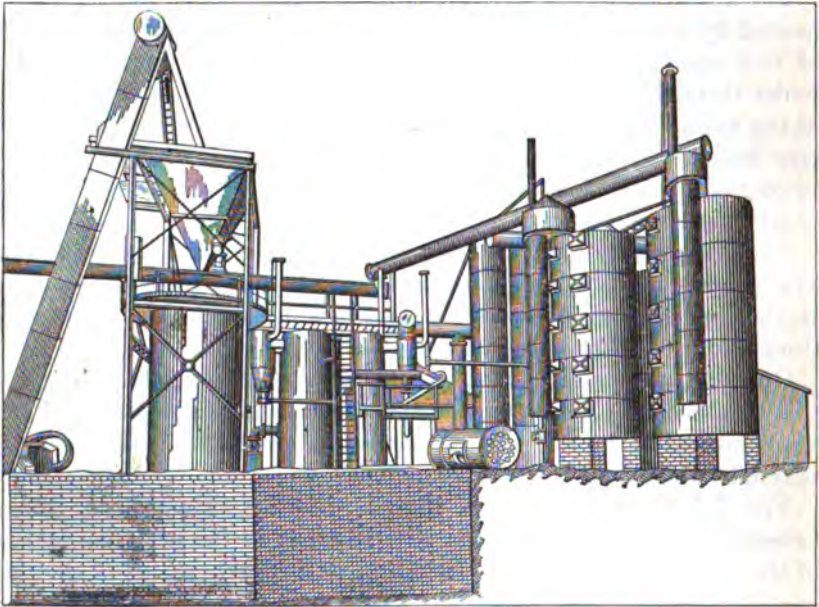


FIGURE 1.—Large gas-producer plant using bituminous coal.

Staffordshire, England. This plant presents so many striking features that the details obtained during a brief inspection are given below:

Number of producers: 8 (2 in reserve).

Make of producers: Mond.

Type of producers: Pressure.

Rated capacity of producers: 2,500 horsepower each.

Number of engines: 2.

Make of engines: Westinghouse.

Type of engines: Vertical, single-acting, 3-cylinder.

Rated capacity of engines: 250 horsepower each.

Hours of operation a day: Twenty-four.

Fuel used: Called slack, but the writer would designate it "run of mine," although it is considerably broken.



A. MOND-GAS PLANT AT DUDLEY PORT, ENGLAND.



B. GAS ENDS OF COMPRESSORS AT MOND PLANT.

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The coal was reported to contain 8 to 9 per cent ash and only 1 to 2 per cent sulphur. The writer understands that the coal formerly bought contained from 13 to 23 per cent ash, but because the cost was too high the company operating the plant finally procured a coal mine, and this yields the better grade of coal indicated. The coal is brought to the works by canal boats, as shown in the general view ^a of the plant given in *A*, Plate I.

The machinery necessary for circulating the acid liquor and water, the boilers for generating steam, and the other auxiliaries of the plant are housed in the buildings shown.

Only a small portion of the gas generated is used in the plant itself, because the purpose of the project, aside from the recovery of sulphate of ammonia, is to distribute producer gas to the manufacturing plants in the neighboring towns, which use it for a variety of commercial purposes. The outgoing gas lines are visible at the extreme right in *A*, Plate I. The main distributing line is 3 feet in diameter.

A better idea of the producers is obtained from the view given in Plate II.

The gas is sent into the mains at a pressure of 5 pounds per square inch. The gas ends of the compressing system are shown in *B*, Plate I. At the time of the writer's visit (August, 1908) there were in use 37 miles of gas mains, the longest single run being $6\frac{1}{2}$ miles.

There is, of course, a ready market for the main product of this plant—sulphate of ammonia. The yield amounts to 80 or 90 pounds per ton of coal fired, as previously stated, and the return is approximately \$2.25 for each ton of coal gasified. Each producer when in operation gasifies on an average during a month 20 tons of coal each twenty-four hours.

The tar by-product, which amounts to considerable in a plant of this size, is also sold—some for roofing, some for paving, some for briquetting, and some for use in distilleries.

The steam required by the plant is generated in Climax boilers, which are so arranged that either coal or producer gas or both may be used as fuel.

The gas, after passing the centrifugal tar extractors, is cleaned by means of a sawdust purifier to insure its freedom from tar before it is turned into the mains, as a plug in the mains underground would be serious. That the method of cleaning has proved effective is shown by the following statement made by Mr. Humphrey in a letter received a few months ago: "Since the plant started four years ago the pressure has never been out of the mains and the supply has never failed."

^a Permission to publish the illustrations of the Mond plant was obtained through the courtesy of H. A. Humphrey, consulting engineer, London.

UTILIZATION OF TAR.

In the pressure-producer plants the tar is extracted mechanically and forms a by-product which is usually of little commercial value; consequently, it becomes a nuisance about the premises and must be drained or carted away. At some places attempts are made to burn it, either in furnaces or in the producers themselves, but in general such attempts have not met with favor, save possibly for certain types of down-draft producers.

One method of utilizing the refuse tar in vogue at certain foreign plants is to mix it with the ash and then to spread the mixture in the yards for walks, roads, etc. There is a limit to this use, however, as the entire yard eventually becomes covered to the maximum depth allowable. As already stated, the tar from the plant at Dudley Port is sold to advantage.

At one European plant visited there is special equipment for handling the tar and preparing it for commercial use. The tarry water runs into a large vat or cistern and most of the tar is separated from the water as the mixture passes over a series of screens. The tar with a reduced percentage of water is then drawn off into a large tank fitted in the bottom with pipes for exhaust steam. By means of the steam the tar is kept warm enough to facilitate its being pumped through a pipe line to tank cars on the near-by railroad. The tar as shipped often contains as much as 20 per cent water.

The average weight of water-free tar recovered in this plant is 90 pounds from each ton of fuel fed to the producer. The water-free tar sells for about \$4.25 a ton, a price which yields a return of approximately 19 cents on each ton of coal fired, not taking into account fixed charges and the labor expense of separating the tar. These items, however, amount to little. The tar is used as a binder for fuel briquets and as a basis for various medicines.

PROGRESS IN DESIGN OF SMALL SUCTION PLANTS.

In Europe, the demand for small power plants to use bituminous coal is quite as evident as in this country. The leading manufacturers of gas producers are all working on suction plants for bituminous coal, and each apparently feels that his model is the only successful one. But each manufacturer places special restrictions on the kinds of fuel that may be used. One manufacturer states that the bituminous coal to be used in the suction plant of his design must be of good quality, low in ash (say 8 to 10 per cent as a maximum) and non-caking, and must not contain more than 7 per cent tar. Another manufacturer makes the statement that for other than anthracite coals the users of his producers confine their attention to short-flaming, low-ash fuels that yield little tar and do not cake.



GAS GENERATORS AT MOND PLANT.

PLANT AT A TOWN IN GERMANY.

Although the writer saw several of these suction plants especially designed for bituminous coal, he found only one in actual operation. This was a double-zone plant of 250 horsepower supplying gas to a 150-horsepower horizontal single-acting twin engine. The coal used was reported to contain 25 per cent ash and 2 per cent sulphur. It was claimed that so long as the coal was noncaking such high ash content gave no trouble. In this installation it was imperative that only coals possessing certain qualities should be used. To comply with these limitations it was often necessary to procure three varieties of coal and to mix them in the proper proportion. All fine coal was sifted and the dust was thrown out to prevent matting. With such restrictions imposed it is difficult to see how similar plants can meet commercial demands. All the makes of suction plants for bituminous coal that were inspected were of the double-zone design.

USE OF LOW-GRADE COAL.

The general situation in Europe regarding the use of low-grade bituminous coal is not much unlike the situation here. For the most part the better-grade coals are being used and the poorer grades are being left in the mines. It is true, however, that this condition is recognized abroad as a serious one, and there is deep interest in the possible utilization of low-grade fuels in the gas producer.

In discussing the matter with the writer, one of the best informed producer-gas experts of England expressed the opinion that the proper way of beginning investigations relating to the general development of the field for producer gas was to take up the utilization of low-grade fuels. He said that such utilization was as important in England as in America, and regretted that more active work along this line had not been undertaken in the former country. He also said that plants for low-grade fuels will eventually be perfected by working on a basis different from the present, for he believed many radical departures from the practice of to-day must be made to get the proper solution of the problem.

By careful inquiry the writer was able to learn of only two plants that are endeavoring to use low-grade coal extensively. One of these plants at the time of his visit was still regarded as in the experimental stage, though it had been erected a year before and had given excellent satisfaction for several months. This plant was of the Mond by-product type, with details modified to suit the peculiar requirements of the low-grade high-ash material used as fuel. This material consisted of a mixture of three varieties of refuse, namely, small-size waste material from the principal dump of the mine, large pieces of clay and rock that when broken showed traces of coal, and fine sludge from the coal washery. The first of these materials was

reported to contain between 60 and 70 per cent ash and the others contained smaller percentages, so that the total ash content of the mixture as charged into the producers was between 50 and 55 per cent. A particularly interesting point noted was that large earthy slabs containing almost no coal yielded a high percentage of nitrogen, in many instances far above that obtained from the coal.

Doctor Frank, who is greatly interested in the possibilities of this type of plant, informed the writer that it was possible to recover 80 per cent of the nitrogen in the fuel. From fuel containing 3 per cent nitrogen he believed the yield of sulphate of ammonia might amount to about 140 kilograms per metric ton (1,000 kilograms) of fuel, and that under these conditions the gas for power could be regarded as costing nothing. From the refuse fuel used in the plant under discussion he stated that the yield of sulphate amounted to 20 or more kilograms (44 or more pounds) and the power equivalent of the gas was 500 horsepower-hours per ton of coal fired.

THE JAHNS "RING" PRODUCER.

The other plant for the utilization of low-grade bituminous coal was the Jahns "ring" gas-producer plant at the Von der Heydt royal colliery, near Saarbrücken, Germany. The details of this are so exceedingly interesting that the following brief extracts have been taken from a description published by Mr. Jahns:

Gas producers with a single combustion chamber as hitherto used are generally replenished from time to time with fresh fuel fed in at the top. Consequently the tar content of the gas from such producers varies between low and high.

The "ring" producer is characterized by several combustion chambers placed side by side that can be connected through passages in such manner (the purpose of the producer being to generate gas that contains little or no tar) that the products of distillation have to pass through large quantities of highly incandescent fuel.

A "ring" producer has at least two chambers, large "ring" producers have three or four chambers, and a large producer plant always has several "rings." Figure 2 illustrates the type of "ring" producer in use at the Von der Heydt royal colliery. Four chambers (I, II, III, IV) form one ring of this plant, which consists altogether of five rings.

At the intersection of the partition walls of the four chambers is a vertical flue or channel (*a*, fig. 2). Passages, *b*, from the top and bottom of each chamber lead into this channel; these passages can be opened or closed separately. In the upper part of the passage *a* is a steam injector that increases the suction through the chambers which are in course of preparation—that is, those which were charged last—in order to expel the volatile matter of the coal as quickly as possible and to bring the charge up to the required heat. The steam from this injector is used for producing gas in the gasifying chamber.

The grate consists of separate bars which can be driven out when fuel containing large quantities of slack is used, so that the refuse can drop freely into the ash pit after the producer is burned out. If fuels containing comparatively little slack and low in ash are to be used, a fixed grate which admits of drawing clinker and ashes at any time is put in. The ash pits are closed by plain doors and slides. The hot refuse remains in the ash pit or on the grate until all its available heat has been used to pre-

heat the air supply and to generate additional steam. Afterwards it is removed in ash trucks.

If a plant is to be used for simultaneously producing gas for heating and for power, it has two separate gas mains and gas exhausters.

No charging is done while the chambers are working in the order in which they were ignited. The chambers are suitably connected with each other through the central flue, and the injector draws the steam and the tar-containing gases of the later kindled producers through the previously kindled producers, which are connected to the gas main. At the same time the air supply, mixed with steam and heated by the refuse in the ash pit and on the grate, is sucked into the chambers at the bottom.

According to the scheme outlined above, the process of gas production in the Jahns "ring" producer is divided normally into two stages, as follows:

- (a) Gas expulsion or distillation, which may be called the preparatory stage.
- (b) Gasification.

In the multichamber producer the duration of the preparatory stage is only a fraction of that of the gasification stage. In a "ring" generator, therefore, several chambers may be prepared one after the other during the gasification period of one chamber—

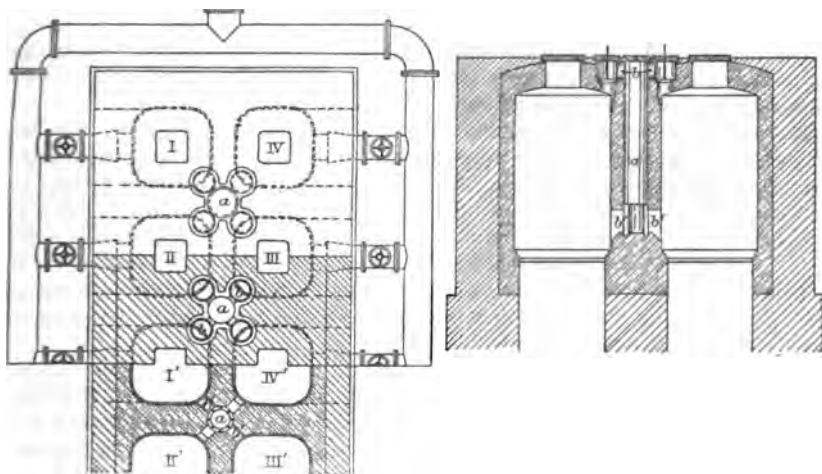


FIGURE 2.—Jahns "ring" producer at the Von der Heydt royal colliery, near Saarbrücken, Germany.

in a three-chamber producer two and in a four-chamber producer three chambers—so that as a rule only one chamber is in preparation when two or three chambers are supplying gas. Therefore, the total working period—that is to say, the combined preparation and gasification time—is the working period of a whole ring, and the capacity of a ring is equal to the capacity of a single chamber multiplied by the number of chambers.

The gases from the chambers in preparation are led into or under the combustion zone of previously kindled chambers, where they are burned or fixed (made permanent) and are drawn off with the gases of these chambers. In the gasification chamber the incandescent fuel decomposes the incoming gases from the preparatory chambers and produces, according to the degree and manner of preparation, gases that are poor in tar and become with advancing gasification still poorer in tar. Nevertheless, the mixture of gases from several chambers remains uniform because the chambers, as they were lighted at different times, are in different stages of gas production.

The power gas contains only such tar vapors as are not volatile at the temperature of the producer, and since they do not decompose in contact with the glowing coal, can be considered as permanent.

If in the course of the process the carbon of the charge in the first chamber is gasified and consumed to such a degree that it can no longer decompose the gases from the preparatory chambers and the steam from the injector, this chamber is shut off from the gas main and its connection to the central channel changed; the steam injector then forces combustion again, the carbon in the charge burns to CO_2 , and the CO_2 formed is decomposed in the other chambers.

This period of complete combustion, or CO_2 formation, is called the "running out" period, and serves to heat the chamber walls to a high temperature for the following preparatory period. At the end of the "running out" period ashes and clinker are dropped into the ash pit; the chamber is then refilled and started as the last chamber of the series.

The emptying and recharging of a chamber take place when it is disconnected from the gas main, and when other chambers are producing gas; gas production is, therefore, neither interfered with nor interrupted. At the same time the steam injector maintains a pressure below atmospheric in the chamber, so that all manipulation can be carried out easily and without danger.

The second chamber to be fired takes up the function of the first one and is treated just as the first was treated; consequently it does not make any difference how many chambers of the ring are connected to the gas main.

The method of operation just described renders possible the direct production from bituminous coal of gas suitable for either heating or power purposes, and not only from pure coals, but also from those containing large amounts of slack.

The power gas is drawn off by means of an exhaustor or a steam injector; the latter may also serve to separate any tar vapors that may be carried over when the producers are not worked with sufficient care. The gas is passed through a scrubber and a saw-dust cleaner to purify and dry it and then passes to the gas holder, if one is used, or to the engines. The exceedingly simple cleaning device requires no attention whatsoever. The calorific value of the gas is constantly shown in a very simple manner—that is, by a small test flame from a jet tapping the pipe leading from the cleaner; the general appearance and color of the flame will enable anybody after a brief experience to estimate the heat value of the gas.

At the Von der Heydt royal colliery a generating plant, consisting of five rings each with four chambers, has been in continuous operation since April, 1904. The results obtained there will certainly confirm all that has been said about the suitability and reliability of the "ring" producer. The generators are charged with the refuse separated from the coals. This refuse contains on an average only 20 per cent of good coal and was formerly thrown on the dump.

The charging must be done at about seven-hour intervals, since the average burning time of a chamber with a charge of about 4 tons when producing power gas is about twenty-six to twenty-eight hours. This time is that of a producer with four chambers in the ring. When fuel of better quality is charged the working time of the chamber under similar conditions is shorter. It will be seen that all the attendance required is to inspect the incandescence of the charge and to clean out the clinkers. The whole working of the plant is so simple that unskilled labor may be employed.

The gas obtained is used under steam boilers and in gas engines. One of the latter, giving an output of 60 horsepower, has been at work without a stop since July, 1904, and the other engine, output 175 horsepower, was started in September, 1904, and has been going ever since.

The gas engines are single-acting, four-cycle engines, the larger one of twin type. The 60-horsepower engine drives the electric light generator and also the ventilators for the producer plant; it is running day and night with varying load. The 175-horsepower engine drives a generator producing multiphase current that supplies an

underground pumping plant (for sixteen hours a day) and the machine shop situated about 300 yards away. It also works a transfer table at the colliery station. The two engines have to take a combined load of 120 horsepower for twenty-four hours each day. So far they have run continuously without any trouble or irregularity.

At the time of his visit in 1908 the writer found the producers in full operation, using roofing slabs that upon casual inspection gave little indication of containing any combustible material. It was said that this fuel averaged over 60 per cent ash, a claim which seemed entirely reasonable. The plant was reported to be using over 100 tons a day of this low-grade fuel.

The producer plant had been enlarged and was not only supplying gas to a number of furnaces, but also to a 1,000-horsepower and a 250-horsepower gas engine. A 500-horsepower engine was being added to the equipment. The engines were direct connected to electric generators. The 10,000-volt current was used for driving the local mine machinery and also for furnishing lights in neighboring towns and power for a street railroad.

USE OF BROWN-COAL BRIQUETS.

The lignite, or brown-coal, briquets made in Germany form an excellent producer-gas fuel and are in common use for the small suction plants, generally of the double-zone type, that are put out by various manufacturers. This brown coal closely corresponds to some varieties of American lignite. In Germany the raw brown coal has proved to be undesirable for producer fuel unless in good-sized pieces similar to those of freshly mined Texas lignite. An interesting sight noted at the plant of one of the manufacturers of these producers was a liberal supply of Texas lignite for experimental purposes.

Suction plants burning lignite apparently require as little attention as those using anthracite, and may need less. In certain localities the writer found operators mixing two kinds of brown-coal briquets, as experience had led them to believe such mixing desirable, but the work was simple, and about all the labor required at some of the plants inspected seemed to be the filling of the hopper with fuel.

An excellent idea of the proportions and general details of one of these plants may be had from figure 3.^a

It will be noted that the gas is taken off from about the middle of the vertical shaft of the producer. When the producer is in operation the fire burns from both the top and bottom zones toward the middle. The relative rate of combustion of the two zones is easily regulated. The covers of the charging hoppers are left open to admit the air necessary for combustion, as is readily seen by referring to

^a The illustrations relating to the use of brown-coal briquets and peat were kindly furnished by Gebrüder Koerting, Koertingedorf, Hamburg, Germany.

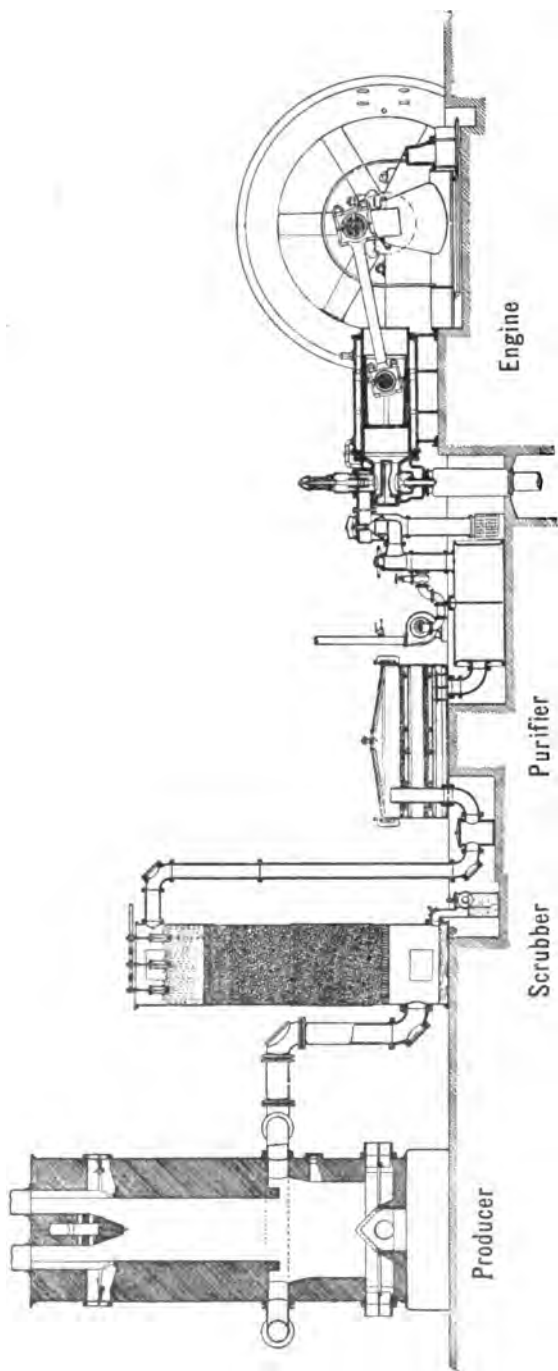


FIGURE 3.—Section of German producer-gas plant burning lignite briquets.

figure 4, which shows the charging floor of one of these plants. Additional air for the lower zone can be readily admitted at the ash pit, as may be seen in the sectional view given in figure 3 or in the view of the main floor of such an installation, figure 5. This type of plant seems exceptionally well adapted to the fuel.

BRIQUET-BURNING GAS PRODUCERS AT FÜRSTENBURG, GERMANY.

An interesting German producer-gas plant using briquets is situated at Fürstenburg on the Oder, about a mile from large brown-coal mines that have been in operation some fifty years. The coal con-

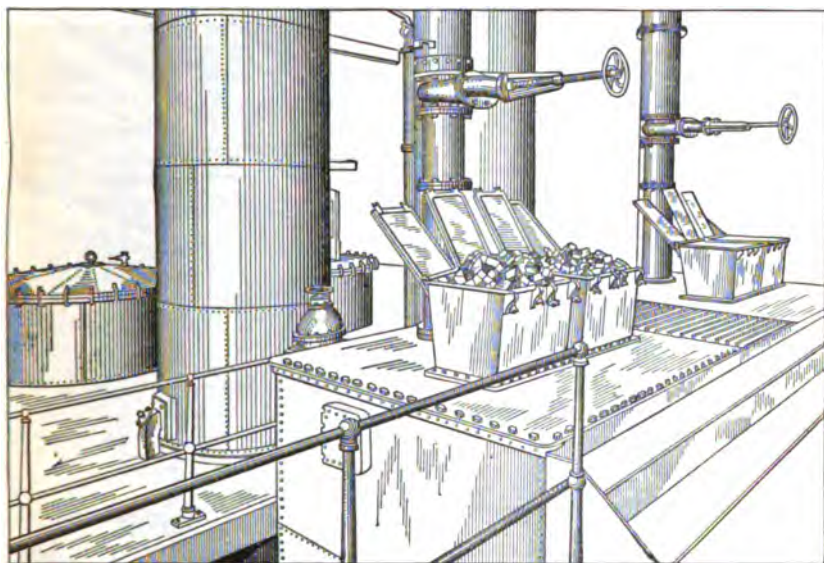


FIGURE 4.—Charging floor of producer plant burning lignite briquets; hoppers open to admit air for combustion.

tains approximately 55 per cent water as it comes from the mine. It is ground, dried until the moisture content is reduced to 11 to 14 per cent, and briquetted without the use of a binder. All the output of the mine is thus utilized. The briquets are delivered continuously from the press to barges lying nearby in a canal. The loading of the barges is done by women, who receive about 30 cents per day.

The briquets are sold chiefly for household use. They have proved of no value for locomotive service, as they fall to pieces when subjected to high temperatures and strong draft and the pieces fly out of the stack. They are not used in large stationary plants because bituminous coal is cheaper. The local producer-gas power plant burning

these briquets comprises the equipment noted below. The power is utilized as indicated:

Number of producers: 2.

Make of producers: Koerting.

Type of producers: Suction.

Rated capacity of producers: 130 horsepower each.

Number of engines: 2.

Make of engines: Koerting.

Type of engines: Single-acting, single-cylinder, 4-cycle, horizontal.

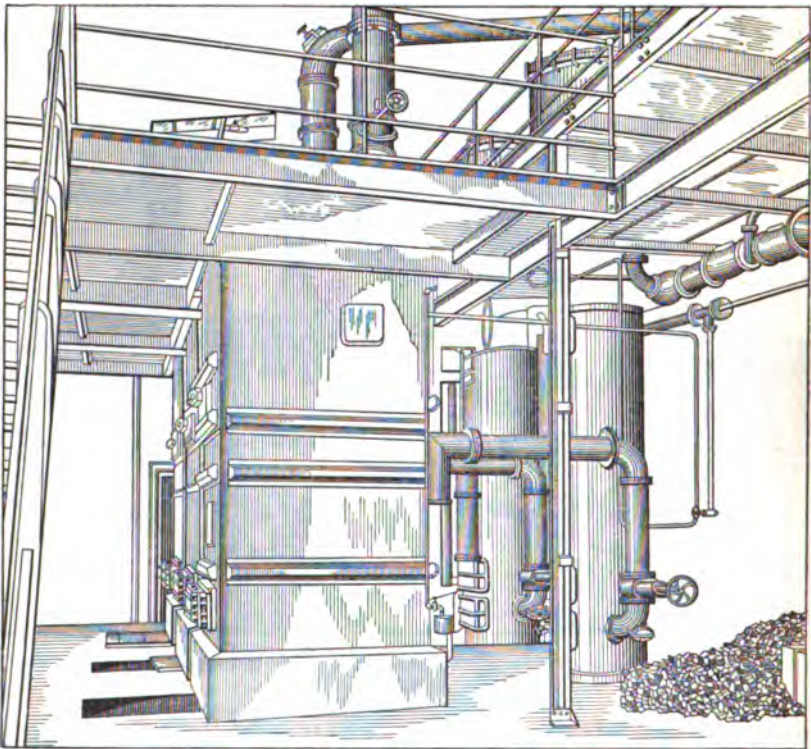


FIGURE 5.—Main floor of plant; doors in ash pit admit air for lower zones.

Rated capacity of engines: 130 horsepower each.

Use of power: Hoisting and other work at the mines, and lighting. (A. C. current, 6,000 volts; transformers at the mine, a mile from the power house.)

Hours of operation per day: One unit twenty-four hours a day for one week; units used alternately.

The engines for this plant are shown in figure 6.

Although an analysis of the lignite mined near the plant described was not available, the following analysis of a lignite used in producers in Austria may be of interest:

Analysis of an Austrian lignite.

Carbon.....	54.56	Moisture.....	22.26
Hydrogen.....	3.97	Ash.....	3.36
Oxygen.....	15.15	Sulphur.....	1.36
Nitrogen.....	.70	Calorific value.....calories..	4,900

USE OF PEAT.

Among the most interesting producer-gas plants in Europe are those utilizing peat as fuel. The peat resources of Europe are extensive and are being rapidly developed. In many countries piles of peat dug for household use may be seen from the railroad trains. In some countries this fuel has thus far been used entirely for domestic purposes, but in others peat forms the main fuel supply for power and manufacturing plants.

The bogs of Ireland have for generations supplied peat for domestic

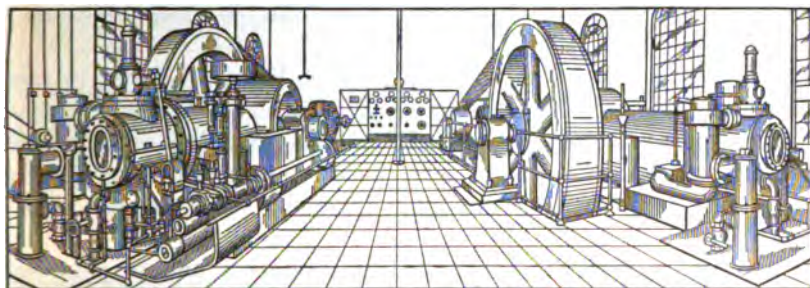


FIGURE 6.—Engines of producer-gas power plant at Fürstenberg, Germany.

fires, but, although the bogs are extensive, the peat has not been used much for generating power. In 1908, however, projects were being considered that involve the erection of extensive power plants in the bogs of central Ireland and the transmission of electric current to Dublin.

Peat has been most largely used in Holland, Austria, Germany, Denmark, Norway, Sweden, Finland, and Russia. In Holland peat has been used for hundreds of years and the bogs have yielded large returns. By the methods pursued, not only is the peat utilized but the bog is left in excellent condition for agriculture.

In Austria and Germany much progress has been made in utilizing peat, especially in the manufacture of peat coke and the making of sulphate of ammonia from peat by means of by-product recovery plants. Doctors Frank and Caro have done much to advance ammonia recovery by their process of gasifying peat in a mixture of air and superheated steam. They find it possible to gasify peat containing 50 to 55 per cent water, thus saving much of the time usually required

for drying. Although much of the peat thus far used in these by-product plants contains only about 1 per cent nitrogen, the returns have been surprisingly good, 40 or more pounds of sulphate of ammonia being obtained from each ton of peat fired.

Doctors Frank and Caro state that from an English peat they obtained 118 pounds of sulphate of ammonia per ton of water-free peat. The gas from each ton of fuel generated all the steam required by the plant and produced power equivalent to 480 horsepower-hours.

Doctor Frank believes that the process will pay with peat containing only 1 per cent nitrogen. The peats of the United States contain a much larger amount of nitrogen than those of Europe, and Doctor Frank is confident that this process, if applied to them, will prove very profitable.

Messrs. Crossley Brothers report that at their works they recovered 134 pounds of sulphate of ammonia from a ton of chemically dry peat containing 2.24 per cent nitrogen. They also report that 1,000 horsepower-hours could be obtained from the gas from each ton of dry peat.

Since the supply of bituminous coal or lignite is large in some countries, the necessity for utilizing the peat resources is not so great in them as in Norway, Sweden, Denmark, and Finland. A recent report shows that Russia leads in the production of peat for generating power, the quantity dug reaching 4,000,000 or 5,000,000 tons a year.

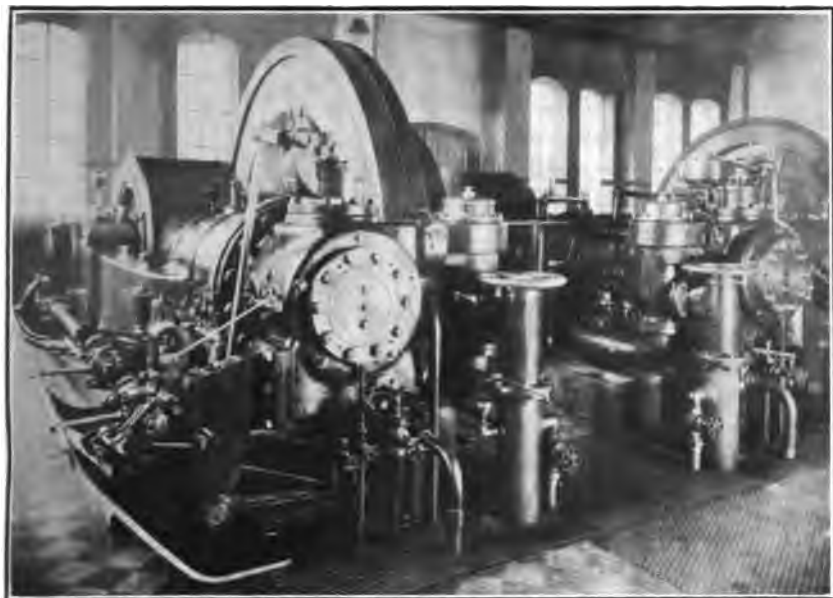
The principal manufacturers of gas producers and producer-gas engines are now putting out double-zone suction producers, designed especially to use peat. These producers burn peat containing 20 to 30 per cent moisture, and seem to work easily and give a rich, clean gas. Figure 7 shows a section of a peat-burning gas-producer outfit that is meeting with general success on the Continent.

These small peat-burning producer plants for generating power are widely used in Europe, although the first plant of this type was installed no longer ago than 1904.

PEAT PRODUCER-GAS PLANT AT SKABERSJÖ, SWEDEN.

The first plant, which stands in the center of a peat bog near Skabersjö, Sweden, is of special interest. Its capacity is only 300 horsepower, and it is situated about 3 miles from the town, to which it supplies the electricity. Half of the plant (150 horsepower) was erected in 1904 and the other half in 1906.

The plant is probably both the first and the smallest producer-gas installation located at a bog and generating high-voltage current for transmission to a point some distance away. In 1908 the plant comprised two suction producers especially designed to burn peat, and rated at 150 horsepower each, and two engines direct connected



A. REAR VIEW OF ENGINES AT SKABERSJÖ, SWEDEN.



B. SIDE VIEW OF ENGINE AT SKABERSJÖ, SWEDEN

to alternating-current three-phase generators, which were running smoothly in parallel at the time of the writer's visit. The 3,000-volt current is transmitted to the town, where it is used during the day for lighting the shops and for driving shop motors and at night for lighting streets and residences. One unit is in operation from 5.30 a. m. to 6 p. m. and the other from 5.30 a. m. to 11 p. m. every day. The charge for residence lighting is 9 cents per kilowatt-hour.

A 35-horsepower peat machine is used for preparing the fuel. This is driven by an electric motor supplied with current from the power plant on the bog. As only 750 tons of dry peat are required per year, there is no attempt to work the plant to its maximum. As there is no difficulty in getting out in the working season, which in this locality is from April 15 to September 1, all the peat needed for a year, only 14 men, local farmers, are employed, and they work as

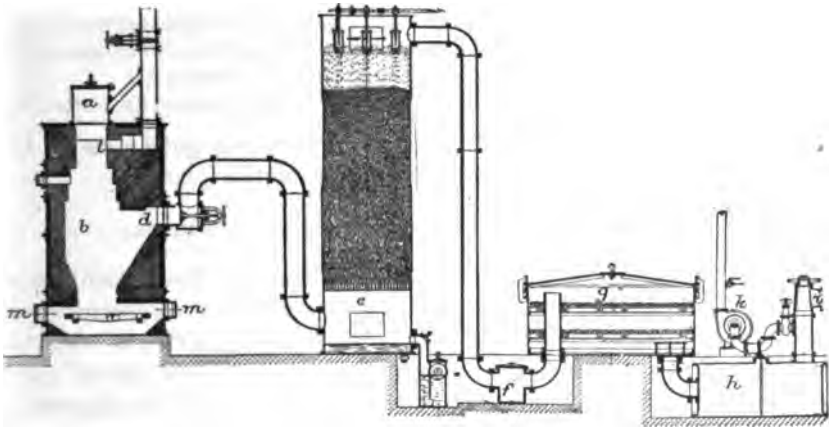


FIGURE 7.—Section of peat producer-gas plant. *a*, *l*, air inlet; *b*, generator chamber; *d*, gas outlet; *m*, air inlets to fire box; *e*, scrubber; *f*, trap; *g*, purifier; *h*, gas collector; *i*, regulator; *k*, aspirator.

little or as much as they please. They receive about 50 cents a day each and get out about 20 tons of peat a day.

Bituminous coal at Skabersjö costs \$3.75 a ton. The dry peat delivered on the platform of the producer plant costs only 80 cents a ton.

The general details of the plant are summarized thus:

Number of producers: 2.

Make of producers: Koerting.

Type of producers: Suction, specially designed for burning peat.

Rated capacity of producers: 150 horsepower each.

Number of engines: 2.

Make of engines: Koerting.

Type of engines: Twin, horizontal, single-acting, 4-cycle.

Rated capacity of engines: 150 horsepower each.

Use of power: Motors in shops and for lighting the shops, streets, and residences at town 3 miles from the plant.

Hours per day: 5.30 a. m. to 6 p. m. for one engine; 5.30 a. m. to 11 p. m. for the other.

Number of men required to operate plant: 3 on day shift (1 for supplying fuel, 1 in the producer room, and 1 in the engine room); 2 at night.

Methods of charging: Charge once an hour.

Kind of fuel and cost: Peat, costing 3 krone (80 cents) a ton at the producer.

Fuel used: 2 kilograms per horsepower-hour.

No fan is used for suction at this plant, the gas being drawn from the producer by the suction strokes of the engines.

The only trouble experienced with the producers has been from the lining, and this has been slight.

It has been found that the plants using this peat operate best with 30 per cent moisture in the fuel; with less the fuel is too dry and steam is required. More than 30 per cent moisture is too much.

The bog is worked by the old type of machine; that is, the peat is shoveled onto the conveyor. The machine is driven by an electric motor, taking current that has passed through a transformer placed on the bog close to the machine.

The average depth of this bog is about 2 meters (6.7 feet), and at the present rate of digging the bog will last about fifty years. It is estimated that 1 cubic meter (1.3 yards) of peat in the bog supplies about 100 to 110 kilograms (220 to 243 pounds) of peat containing 25 to 30 per cent moisture.

The plant was running very smoothly and quietly and required little attention.

The accompanying plate (Pl. III, A, B) gives a good idea of the appearance and arrangement of the engines at this plant.

PEAT PRODUCER-GAS PLANT AT VISBY, SWEDEN.

Another peat-burning plant that is attracting the attention of engineers interested in producer-gas development is at a cement works at Visby, Gothland, Sweden. At the time of the writer's visit the capacity of this plant was about to be increased from 250 to 1,500 horsepower. The accompanying plan (Pl. IV) shows the layout of the installation as it stands to-day.

In 1908 the general details of the plant at Visby were as follows:

Number of producers: 1.

Make of producers: Koerting (for peat).

Type of producer: Suction; fan exhausts gas and forces it to engine.

Rated capacity of producer: 250 horsepower.

Number of engines: 1 twin unit.

Make of engine: Koerting.

Type of engine: Horizontal, single-cylinder, single-acting, twin.

Rated capacity of engine: 250 horsepower.

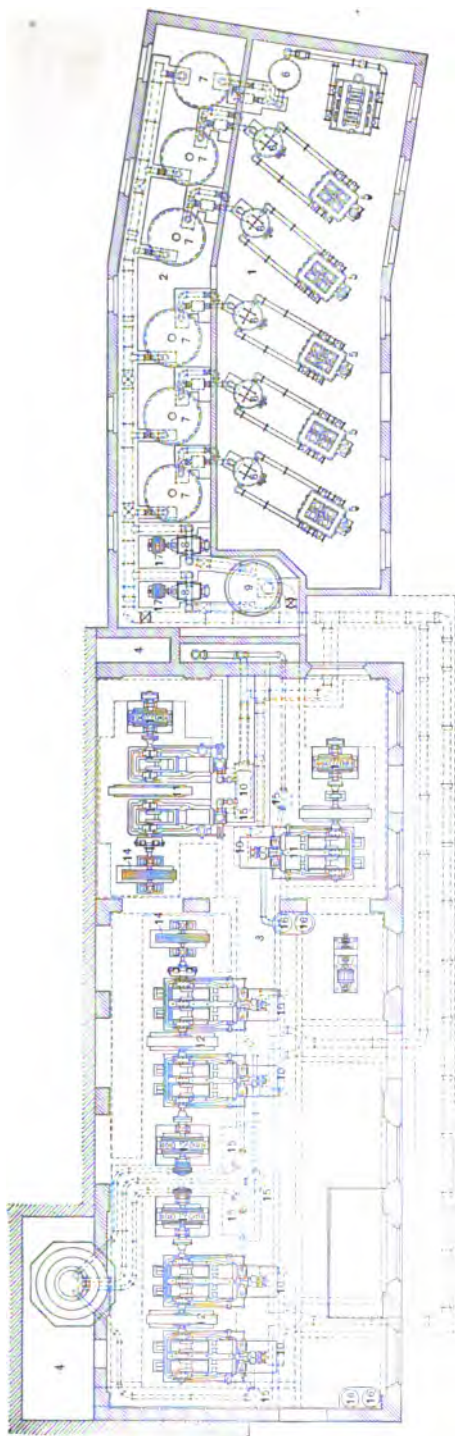
Load carried: Full.

Use of power: Driving machinery in cement works.

Hours of work per day: 10.

Kind of fuel, cost, etc.: Peat, costing, with old methods of collecting, \$1.35 per ton on cars at bog. English anthracite costs nearly \$6 per ton.

The dry peat from the old machine at the bog is more or less broken and varies in size from pieces 12 by 4 by 2 inches to dust. The peat is brought on cars from the bog and dropped into a big concrete storage house, well ventilated by slits in the walls. From this storage house boys take the peat to a crusher, first shaking it out on forks to deposit all small pieces and dust. The crusher



PLAN OF 1,500-HORSEPOWER PRODUCER-GAS PLANT AT CEMENT WORKS, VISBY, SWEDEN.

- | | | |
|-------------------|-----------------------------|-------------------------------|
| 1. Producer room. | 10. Expansion chambers. | 14. Driving wheels. |
| 2. Purifier room. | 11. 250-horsepower engines. | 15. Mufflers. |
| 3. Engine room. | 12. 500-horsepower engine. | 16. Tanks for compressed air. |
| 4. Air filters. | 13. Electric generators. | 17. Electric motors. |
| 5. Producers. | | |

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breaks it into pieces from 1 to 3 inches in diameter, which are carried by a mechanical conveyer to the bins over the producer.

Boys weigh each wheelbarrow load before dumping it into the crusher. During my stay about the plant (about one and one-half hours) nobody went near the producer. The engines were running nicely and no troubles were apparent. There was serious trouble from tar in the engines when the plant was first started, but this had been overcome.

The dry peat usually contains about 25 per cent of moisture, but at times contains 40 per cent.

PREPARATION OF PEAT AT VISBY.

The working season at Visby is from April 15 to August 8, and all the peat should be dry and in the storage bins by October 1. At the time of the writer's visit hundreds of tons of peat in various stages of drying were scattered over the bog.

The old peat machine, installed in 1902, is of 38 horsepower; that is, the boiler and engine which operate it have about that capacity. The peat is shoveled onto the conveyer, and men with a series of knives cut the strips from the mixer into the required lengths. The peat blocks are irregular in size and always break more or less during drying. Originally 26 men were required for an output of 30 tons of "dry" peat (containing 25 per cent moisture) in a ten-hour day; later the number of men was reduced to 23. The machine cost \$3,375.

The new machine, installed in the spring of 1908, is the first of its type, and when seen was, of course, still somewhat experimental. It is of 42-horsepower capacity, cost \$7,155, and with a crew of 10 men turns out 60 tons of "dry" peat in a ten-hour day. The crew comprises 1 engineer and 1 helper, 1 digger (at the buckets), 4 field-press men, 2 track layers, and 1 trolley man who couples and guides the cars.

With the new machine the peat is dug by a bucket dredge and conveyer, passes through shredding and mixing devices, and runs into the cars as a pasty mass. It is then taken to the drying field, dumped into a frame on the ground, leveled with hoes, rolled and divided into bricks. The longitudinal divisions are made by blades on the roller, the transverse are made by a 3-disk cutter pushed by a man. The bricks measure about 8 or 10 inches long, 4 inches wide, and 2 inches thick and weigh (when containing 25 per cent moisture) 1.75 pounds each.

The average depth of the bog is approximately 6.7 feet, and the output of dry peat is about 370 pounds per cubic yard of peat from the bog.

About thirty days are required for drying—two weeks before piling and two weeks after. The women who pile the peat receive 12.5 cents per 1,000 pieces, and their average wage is 47.5 cents to

54 cents per day. The cables that drag the cars of peat about the field are long and break frequently at the bends made by the car grips. With the old systems of work, horses are used to handle the cars.

To load the peat at the railroad, the small cars are hauled up an incline and dumped into the railroad cars. As the hauling requires the attention of three or four men, it is planned to introduce gas-engine haulage. The fine broken peat is used in the steam boiler and makes an excellent fuel.

USE OF BLAST-FURNACE OR COKE-OVEN GAS.

The writer has made no attempt in this brief discussion of what he saw to touch upon the use of either blast-furnace gas or coke-oven gas in internal-combustion engines. Very large power plants using these fuels are to be found throughout Europe, and the adaptation of such waste gases for generating power seems to be looked upon with favor in the principal manufacturing centers. Progress in this direction has been rapid within the past few years.

PUBLICATIONS ON FUEL TESTING.

The following publications, except those to which a price is affixed, can be obtained free by applying to the Director of the Bureau of Mines, Washington, D. C. The priced publications can be purchased from the Superintendent of Documents, Government Printing Office, Washington, D. C.

PUBLICATIONS OF THE UNITED STATES GEOLOGICAL SURVEY.

BULLETIN 261. Preliminary report on the operations of the coal-testing plant of the United States Geological Survey at the Louisiana Purchase Exposition, in St. Louis, Mo., 1904; E. W. Parker, J. A. Holmes, M. R. Campbell, committee in charge. 1905. 172 pp. 10 cents.

PROFESSIONAL PAPER 48. Report on the operations of the coal-testing plant of the United States Geological Survey at the Louisiana Purchase Exposition, St. Louis, Mo., 1904; E. W. Parker, J. A. Holmes, M. R. Campbell, committee in charge. 1906. In three parts. 1492 pp., 13 pls. \$1.50.

BULLETIN 290. Preliminary report on the operations of the fuel-testing plant of the United States Geological Survey at St. Louis, Mo., 1905, by J. A. Holmes. 1906. 240 pp. 20 cents.

BULLETIN 323. Experimental work conducted in the chemical laboratory of the United States fuel-testing plant at St. Louis, Mo., January 1, 1905, to July 31, 1906, by N. W. Lord. 1907. 49 pp. 10 cents.

BULLETIN 325. A study of four hundred sterling tests made at the fuel-testing plant, St. Louis, Mo., 1904, 1905, and 1906, by L. P. Breckenridge. 1907. 196 pp. 20 cents.

BULLETIN 332. Report of the United States fuel-testing plant at St. Louis, Mo., January 1, 1906, to June 30, 1907; J. A. Holmes, in charge. 1908. 299 pp. 25 cents.

BULLETIN 334. The burning of coal without smoke in boiler plants; a preliminary report, by D. T. Randall. 1908. 26 pp. 5 cents. (See Bull. 373.)

BULLETIN 336. Washing and coking tests of coal and cupola tests of coke, by Richard Moldenke, A. W. Belden, and G. R. Delamater. 1908. 76 pp. 10 cents.

BULLETIN 339. The purchase of coal under government and commercial specifications on the basis of its heating value, with analyses of coal delivered under government contracts, by D. T. Randall. 1908. 27 pp. 5 cents. (See Bull. 428.)

BULLETIN 343. Binders for coal briquets, by J. E. Mills. 1908. 56 pp.

BULLETIN 362. Mine sampling and chemical analyses of coals tested at the United States fuel-testing plant, Norfolk, Va., in 1907, by J. S. Burrows. 1908. 23 pp. 5 cents.

BULLETIN 363. Comparative tests of run-of-mine and briquetted coal on locomotives, including torpedo-boat tests and some foreign specifications for briquetted fuel, by W. F. M. Goes. 1908. 57 pp., 4 pls.

BULLETIN 366. Tests of coal and briquets as fuel for house-heating boilers, by D. T. Randall. 1908. 44 pp., 3 pls.

BULLETIN 367. Significance of drafts in steam-boiler practice, by W. T. Ray and Henry Kreisinger. 1909. 61 pp.

BULLETIN 368. Washing and coking tests of coal at Denver, Colo., by A. W. Belden, G. R. Delamater, and J. W. Groves. 1909. 54 pp., 2 pls.

BULLETIN 373. The smokeless combustion of coal in boiler plants, by D. T. Randall and H. W. Weeks. 1909. 188 pp. 20 cents.

BULLETIN 378. The purchase of coal under government specifications, by J. S. Burrows. 1909. 44 pp. 10 cents. (See Bull. 428.)

BULLETIN 382. The effect of oxygen in coal, by David White. 1909. 78 pp., 3 pls.

BULLETIN 385. Briquetting tests at the United States fuel-testing plant, Norfolk, Va., 1907-8, by C. L. Wright. 1909. 41 pp., 9 pls.

BULLETIN 392. Commercial deductions from comparisons of gasoline and alcohol tests on internal-combustion engines, by R. M. Strong. 1909. 38 pp.

BULLETIN 393. Incidental problems in gas-producer tests, by R. H. Fernald, C. D. Smith, J. K. Clement, and H. A. Grine. 1909. 29 pp.

BULLETIN 402. The utilization of fuel in locomotive practice, by W. F. M. Goes. 1909. 28 pp.

BULLETIN 403. Comparative tests of run-of-mine and briquetted coal on the torpedo boat *Biddle*, by Walter T. Ray and Henry Kreisinger. 1909. 49 pp.

BULLETIN 412. Tests of run-of-mine and briquetted coal in a locomotive boiler, by Walter T. Ray and Henry Kreisinger. 1909. 32 pp.

BULLETIN 416. Recent development of the producer-gas power plant in the United States, by R. H. Fernald. 1909. 82 pp., 2 pls. 15 cents.

BULLETIN 428. The purchase of coal by the Government under specifications, with analyses of coal delivered for the fiscal year 1908-9, by G. S. Pope. 80 pp. 10 cents.

PUBLICATIONS OF THE BUREAU OF MINES.

BULLETIN 1. The volatile matter of coal, by H. C. Porter and F. K. Ovitiz. 1910. 56 pp., 1 pl.

BULLETIN 2. North Dakota lignite as a fuel for power-plant boilers, by D. T. Randall and Henry Kreisinger. 1910. 42 pp., 1 pl.

BULLETIN 3. The coke industry of the United States as related to the foundry, by Richard Moldenke. 1910. 32 pp.

Bulletin 5

DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

JOSEPH A. HOLMES, DIRECTOR

WASHING AND COKING TESTS
OF COAL

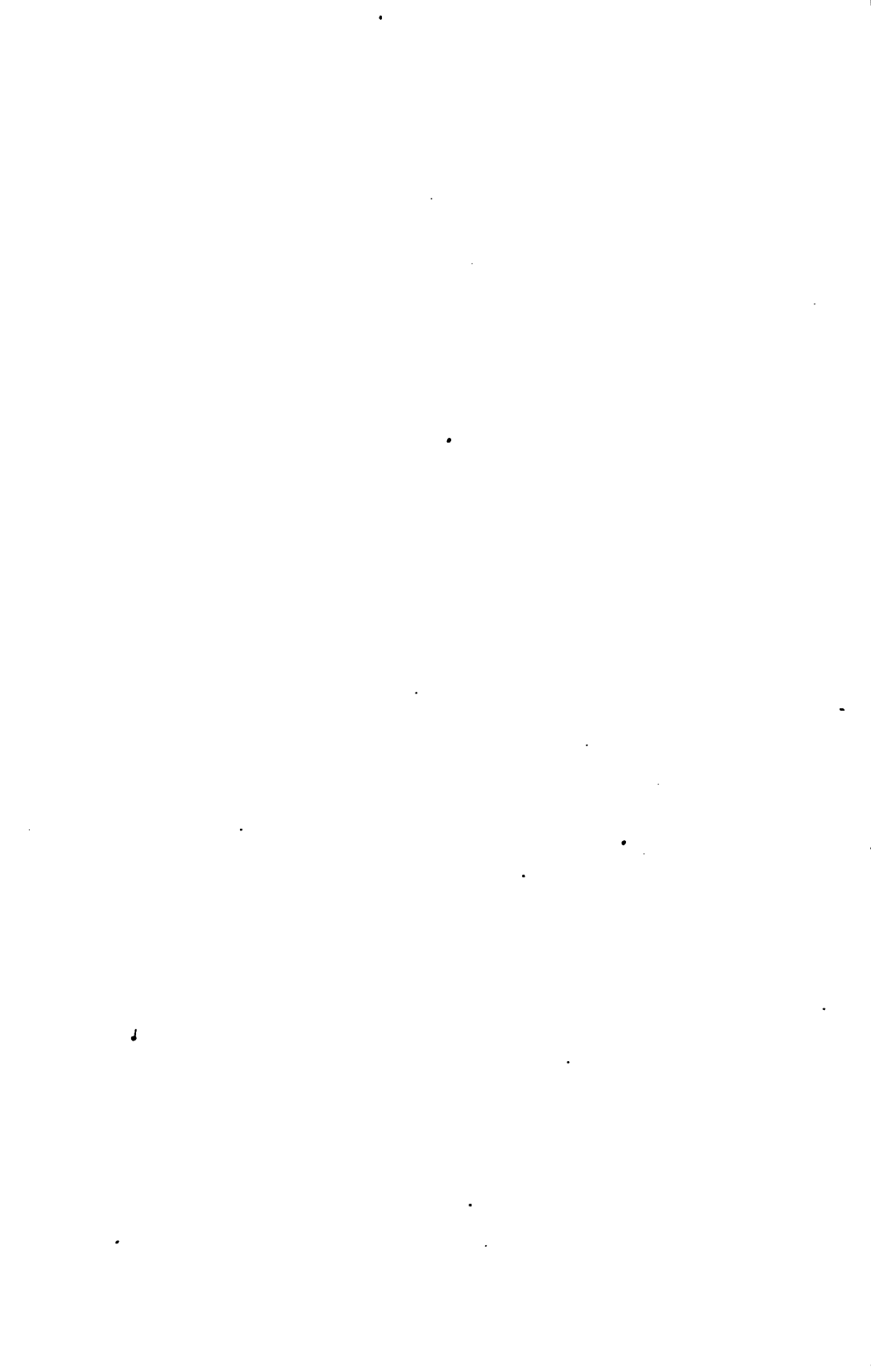
AT THE FUEL-TESTING PLANT, DENVER, COLO.
JULY 1, 1908, TO JUNE 30, 1909

BY

A. W. BELDEN, G. R. DELAMATER, J. W. GROVES
AND K. M. WAY



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WASHING AND COKING TESTS OF COAL AT DENVER, COLO., JULY 1, 1908, TO JUNE 30, 1909.

By A. W. BELDEN, G. R. DELAMATER, J. W. GROVES, and K. M. WAY.

INTRODUCTION.

By A. W. BELDEN.

This bulletin deals with washing and coking tests of coal made at the government fuel-testing plant in Denver, Colo., between July 1, 1908, and June 30, 1909. The tests formed part of the investigation of mineral fuels carried on by the United States Geological Survey under the authority conferred by acts of Congress, and were a continuation of the work started at St. Louis, Mo., during the Louisiana Purchase Exposition. They were made at Denver because of the railroad facilities and central location of that city with respect to coal fields from which little or no coal had been received at the St. Louis plant. The general purpose of the investigations was (1) to determine the possibility of improving the quality of various coals by washing, so as to make them available for the production of coke, and (2) to determine the feasibility of making coke in beehive ovens from these coals. The results of the tests made at Denver are published by the Bureau of Mines because the law creating the Bureau transferred to it the testing of fuels as carried on by the United States Geological Survey.

The Denver plant began operations November 19, 1907. Between that date and March 21, 1908, when work was temporarily suspended, 34 washing tests and 52 coking tests were made on 14 carload samples of coal, representing deposits in Colorado, Montana, Utah, and New Mexico. The results of these tests have been described in a previous bulletin.^a

After the completion of this first series of tests it was decided, because coals representing deposits in a number of important fields had not been received for test, to continue the work long enough to gather the desired information regarding the additional coals and to compare the results of washing and coking tests with those on coals previously tested.

^aBelden, A. W., Delamater, G. R., and Groves, J. W., Washing and coking tests of coal at the fuel-testing plant, Denver, Colo., July 1, 1907, to June 30, 1908: Bull. U. S. Geol. Survey No. 368, 1909, 54 pp.

The plant resumed operations August 21, 1908, and was operated continuously until January 2, 1909. During this period 20 cars of coal were received from 19 places, representing five States and one Territory, as follows: Colorado, 11 cars; Washington, 3 cars; Illinois, 2 cars; Kansas, 1 car; Montana, 1 car; New Mexico, 1 car; and Wyoming, 1 car. On these 20 samples the washery section made 49 tests and the coking section 69 tests. The chemical laboratory made 532 analyses, entailing 3,253 different determinations.

The washery tests were conducted in practically the same manner as those described in Bulletin 368, except for some minor changes in and additions to the equipment, fully described under the discussion of the tests on succeeding pages.

The coking tests were carried on in precisely the same way as was outlined in Bulletin 368. Of the 22 samples tested, all but five produced good coke in the beehive oven by proper treatment. The yield of coke from a number of coals was materially greater than that obtained by concerns coking the same coals in a commercial way. For one coal this difference was over 4 per cent. Such an increase is of much greater importance than it might seem to be at first thought. Consider, for instance, a plant producing 20,000 tons of coke a month. If the yield is 60 per cent of the coal charged to the oven when it might be made 64 per cent, the net yearly loss, with coke selling at \$3.50 per ton, f. o. b. cars at ovens, would amount to \$55,986.

FIELD WORK.

By J. W. GROVES and K. M. WAY.

TEST REQUIREMENTS.

The stipulations under which coal was submitted for test and the methods of inspecting the coal and taking the mine samples were the same as for the previous tests at Denver, and as for the years 1905 and 1906, when the plant was at St. Louis, Mo. These stipulations were as follows:

1. The coal must be furnished to the testing plant free of cost to the Government.
2. The coal must be loaded under the supervision of one of the inspectors employed for that purpose, who shall at the same time be allowed to visit the working places in the mine to secure samples for analysis.
3. When it is possible to do so, the coal should be loaded in box cars and shipped under seal. Lignites must always be shipped in this way.
4. Where the market requires screened coal, this grade will be accepted for tests. The selection of coal is always to be under the direct control of the representative of the testing plant.
5. Where one of the problems involved is the better utilization of slack coal, a carload of slack may be accepted for testing purposes.

6. As soon as possible after the tests are completed, a brief statement of the results will be furnished to parties supplying the coal, for their information, but this must not be made public until the results are made public by the Geological Survey.

7. Everyone interested in any particular test or in the general operation of the plant is invited to be present at any time, but the official record of the tests will not be given out except as indicated in the preceding paragraph.

8. In view of the probability of receiving applications from two or more operators working the same bed of coal in the same locality, the right is reserved to accept but one such application, in order that unprofitable duplication of results may be avoided.

INSPECTION PROCEDURE.

In collecting samples the inspector made certain that the coal loaded for testing represented the average product of the mine, and, if screened coal was to be sent, that the size selected was commercially available and adapted to the purpose for which it was to be tested. Therefore, he examined the screens and observed the method generally employed in separating slate and other impurities from the coal while it was being loaded in the railroad car. He then entered the mine and carefully noted the character of the bed, the method of mining, and the care taken by the miners to separate the shale partings and sulphur balls from the coal while loading mine cars. While the sample was being loaded into the railroad car the inspector stood inside the tipple and permitted the slate pickers to discard only such material as was thrown off in the usual practice and as was consistent with the customary speed of loading. The material thrown out was afterwards examined by the inspector to determine why it was considered detrimental to the fuel value of the coal.

MINE SAMPLING.

In mine sampling the object of the inspector was to secure a sample that would, as nearly as possible, represent run-of-mine coal. He first made a careful study of the coal throughout the mine and observed what parts of the coal bed were discarded by the miner. In an ordinary mine two or more places where the coal was of average development were then selected. These places were usually at widely separated points in the part of the mine from which most of the coal was being shipped. The selected face was cleaned for 5 feet or so, and insecure pieces of the roof were taken down. The sampler then spread a waterproof blanket close up to the face and made a perpendicular cut from floor to roof, including in the sample everything but the parts of the bed discarded by the miner, and cutting not less than 5 pounds of coal per foot in height—that is, a sample weighing not less than 30 pounds from a 6-foot seam. Great care was exercised in cutting included shale or other partings the full

width and depth of the groove in order to preserve the proper proportion of coal and extraneous matter. A detailed record was made of the section of the bed from top to bottom, every perceptible parting and variation being noted. The parts of the bed not included in the sample were clearly shown in this record.

The cuttings were at once weighed and then sifted through a screen with a half-inch mesh. The remaining lumps were broken up on a portable bucking board, and this screening and breaking was continued until the entire sample passed through the screen.

The sample was then mixed by two men grasping the corners of the blanket and rolling the sample diagonally by raising one corner of the blanket at a time. When the larger pieces were seen to be evenly distributed throughout the mass, the sample was quartered, two opposite quarters discarded, and the remainder mixed as before. If the sample was still too bulky to be conveniently handled it was again mixed and quartered.

The remaining material was spread in a circular mass about 2 inches deep on the blanket, and a small trowel was used to fill a sample can with alternate sections of the sample taken from the circumference to the center of the mass around the entire circle. The can was closed and hermetically sealed with electrical insulating tape and the weight noted. This weight showed what proportion of the original sample was sent to the laboratory.

The entire process of sampling was carried on as rapidly as possible. The maximum time for cutting and preparing a large sample was about one hour. Although it is known that rapid changes take place in the moisture content of some coals, it is reasonable to assume that where the sampling is quickly done in the atmosphere to which the native coal is exposed there is probably only a slight gain or loss of moisture while the sample is being broken up and quartered.

DESIGNATION OF SAMPLES.

The samples of coal sent to the Denver testing plant are designated in this bulletin Denver No. 15, Denver No. 16, and so on, consecutively, continuing the series of numbers used in Bulletin 368, "Denver" being placed before each number to distinguish the samples from those sent to St. Louis, Mo.; Norfolk, Va.; and Pittsburg, Pa. The samples are designated by these numbers throughout the tests and in the published results.

In like manner the laboratory numbers of the analyses are followed by the letter D to show that the analyses were made at Denver. For Denver Nos. 29 and 30 analyses of mine samples previously taken and analyzed at Pittsburg are used.

SAMPLES TESTED AT DENVER.

The following is a complete list of the samples tested at Denver from August 21, 1908, to January 2, 1909:

List of coals tested at Denver, Colo.

[All samples inspected by either J. W. Groves or K. M. Way.]

Designation of sample.	Kind of fuel.	Name of bed.	Locality.	Railroad.	Days exposed to weather.
Denver No. —					
15.....	Bituminous, run of mine.	No. 6.....	Sesser, Franklin County, Ill.	Chicago, Burlington and Quincy.	13
16.....	do.	Sopris.....	Sopris, Las Animas County, Colo.	Colorado Southern....	13
17.....	do.	Walsen.....	Bowen, Las Animas County, Colo.	do.....	9
18.....	do.	Primer.....	Primer, Las Animas County, Colo.	Colorado and Wyoming.	17
19.....	do.	Nos. 4, 5.....	Taylor, King County, Wash.	Columbia and Puget Sound.	23
20.....	do.	No. 3.....	Carbonado, Pierce County, Wash.	Northern Pacific.....	54
21.....	Bituminous, run of mine, 1-inch screenings.	Roslyn.....	3 miles west of Roslyn, Kittitas County, Wash.	do.....	32
22.....	Bituminous, run of mine.	(?)	Lombard, Broadwater County, Mont.	do.....	19
23.....	do.	Cherokee.....	3 miles west of Frontenac, Crawford County, Kans.	Atchison, Topeka and Santa Fe.	21
24.....	Refuse.....		Starkville, Las Animas County, Colo.	do.....	12
25.....	Bituminous, 1-inch screenings.	Cameron.....	1½ miles southwest of Rugby, Las Animas County, Colo.	Colorado and Southern.	14
26.....	Subbituminous, run of mine.	Lower seam.	Lafayette, Boulder County, Colo.	Colorado and Southern.	13
27.....	Bituminous, run of mine.	Berwind.....	1 mile west of Berwind, Las Animas County, Colo.	Colorado and Southern.	25
28.....	Bituminous, 4-inch, run of mine.	(?)	Cameo, Mesa County, Colo.	Denver and Rio Grande and Colorado Midland.	10
29.....	Bituminous, run of mine.	Upper.....	Coal Basin, Pitkin County, Colo.	Crystal River.....	20
30.....	do.	Lower.....	do.....	do.....	16
31.....	do.	Carthage.....	Carthage, Socorro County, N. Mex.	New Mexico Midland..	27
32.....	do.	Yampa.....	Oak Creek, Routt County, Colo.	Denver, Northwestern and Pacific.	12
33.....	do.	No. 7.....	Sweetwater, Sweetwater County, Wyo.	Union Pacific.....	13

DETAILED DESCRIPTIONS OF SAMPLES.

DENVER No. 15.

Bituminous coal from No. 6 bed at Sesser, Franklin County, Ill., on the Chicago, Burlington and Quincy Railroad, was designated Denver No. 15.

One sample shipped from this place consisted of 44 tons of run-of-mine coal, which was used in making washing tests 232, 233, 234, and 235, and coking tests 247, 248, 249, 250, and 251.

A second sample from this place, which consisted of 33 tons of 1½-inch screenings and was designated Denver No. 15-B, was used in making washing tests 236, 238, 239, and 240, and coking tests 252, 254, 256, and 258.

Two mine samples were taken for chemical analysis; sample 477-D was taken 980 feet northwest of the shaft, where the coal measured as shown in section A; sample 478-D was taken 1,220 feet north of the shaft, where the coal measured as shown in section B.

Section A (sample 477-D).			Section B (sample 478-D).		
	Ft.	in.		Ft.	in.
Coal.....	7		Coal.....	3	10
Rash.....	1		Shale.....		½
Coal.....	3	9	Coal.....		8
Shale ^a	1½		Blue band ^a		1½
Coal.....	8		Coal.....	1	10
Blue band ^a	1½			6	6
Coal.....	1	7			
	6	11			

Chemical analyses of Denver No. 15 coal.

	Mine samples.						Car samples.					
	477-D.			478-D.			496-D.			495-D.		
	Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.
Air-drying loss.....	1.20	1.10	3.60	3.30
Moisture.....	8.05	9.15	7.57	8.59	4.69	8.12	5.49	8.61
Prox. Volatile matter.....	35.40	34.98	38.50	32.51	32.15	35.17	34.46	37.51	34.32	33.19	36.30
Fixed carbon.....	47.85	47.27	52.03	51.60	51.03	55.63	50.60	48.79	53.10	48.42	46.82	51.22
Ash.....	8.70	8.60	9.47	8.32	8.23	9.00	8.95	8.63	9.39	11.77	11.36	12.48
Ult. Sulphur.....	1.00	.99	1.09	1.30	1.29	1.41	1.17	1.13	1.23	.98	.95	1.04
Hydrogen.....	5.26	4.73	5.15	4.58
Carbon.....	60.44	72.31	64.93	71.05
Nitrogen.....	1.33	1.45	1.36	1.49
Oxygen.....	17.21	10.89	16.23	9.39
Caloric value:												
Determined—												
Calories.....	6,805	6,730	7,363	6,952	6,702	7,295	6,656	6,436	7,042
B. t. u.....	12,114	12,114	13,253	12,514	12,064	13,131	11,981	11,585	12,676
Calculated from ultimate analysis—												
Calories.....	6,465	7,036	6,343	6,941
B. t. u.....	11,637	12,665	11,417	12,493

DENVER No. 16.

Bituminous coal from the Sorpris or Cameron bed at Sorpris, Las Animas County, Colo., on the Colorado and Southern Railway, was designated Denver No. 16.

^a Not included in sample.

One sample shipped from this place consisted of 50 tons of run-of-mine coal and was used in making washing tests 242, 247, and 249, and coking tests 259, 260, 266, 268, and 271.

Two mine samples were taken for chemical analysis. Sample 479-D was taken 7,000 feet southwest of the slope, where the coal measured as shown in section A; sample 485-D was taken 8,800 feet southeast of the slope, where the coal measured as shown in section B.

Section A (sample 479-D).			Section B (sample 485-D).		
	Ft.	in.		Ft.	in.
Coal.....	3	11	Bone coal ^a	9½	
			Coal.....	1	2
			Hard coal (good).....	6	
			Coal.....	2	
			Bone coal ^a	1½	
			Coal.....	1	8
				4	5

Chemical analyses of Denver No. 16 coal.

		Mine samples.						Car sample (536-D).		
		479-D.			485-D.					
		Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.
Air-drying loss.....		2.10			1.40			1.10		
Prox.	Moisture.....	.31	2.40		.51	1.90		1.26	2.35	
	Volatile matter.....	29.92	29.29	30.01	31.23	30.79	31.39	31.03	30.69	31.43
	Fixed carbon.....	57.05	55.96	57.25	57.23	56.43	57.52	53.39	52.80	54.07
	Ash.....	12.72	12.45	12.74	11.03	10.88	11.09	14.32	14.16	14.50
Ult.	Sulphur.....	.90	.88	.90	.70	.69	.70	.65	.64	.66
	Hydrogen.....								4.63	4.48
	Carbon.....								71.12	72.83
	Nitrogen.....								1.04	1.07
Oxygen.....									8.41	6.46
Calorific value:										
Determined--										
Calories.....					7,610	7,503	7,648	7,183	7,104	7,275
B. t. u.....						13,506	13,766	12,929	12,787	13,095
Calculated from ultimate analysis--										
Calories.....									6,995	7,163
B. t. u.....									12,591	12,894

DENVER No. 17.

Bituminous coal from the Walsen bed of coal at Bowen, Colo., on the Colorado and Southern Railway, was designated Denver No. 17.

One sample shipped from the bed consisted of 33 tons of run-of-mine coal and was used in making washing tests 237, 241, and 244, and coking tests 253, 255, 257, and 261.

Two mine samples were taken for chemical analysis. Sample 480-D was taken 2,000 feet north of the drift opening, where the coal

^a Not included in sample.

measured as shown in section A; sample 481-D was taken 2,600 feet northwest of the drift opening, where the coal was measured as shown in section B.

Section A (sample 480-D.)			Section B (sample 481-D.)		
	Ft.	in.		Ft.	in.
Coal.....	3		Coal.....	11	
Bone coal ^a		3	Bone coal ^a		6
Coal.....	1	10	Coal.....	2	6
	5	1	Bone coal ^a		2
			Coal.....		11
				5	0

Chemical analyses of Denver No. 17 coal.

		Mine samples.						Car sample (484-D).		
		480-D.			481-D.					
		Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.
Prox.	Air-drying loss.....	1.70			2.20			0.50		
	Moisture.....	.78	2.47		.79	2.97		1.41	1.90	
	Volatile matter.....	34.24	33.66	34.51	33.23	32.50	33.50	31.52	31.26	31.96
	Fixed carbon.....	53.91	52.99	54.33	51.69	50.55	52.09	49.16	48.92	49.88
	Ash.....	11.07	10.88	11.16	14.29	13.98	14.41	17.91	17.82	18.16
	Sulphur.....	.83	.82	.84	.66	.65	.67	.68	.68	.69
	Hydrogen.....								4.63	4.49
	Carbon.....								67.82	69.13
	Nitrogen.....								1.18	1.20
	Oxygen.....								7.87	6.33
Ult.	Calorific value:									
	Determined—									
	Calories.....				7,092	6,936	7,148	6,737	6,703	6,833
	B. t. u.....					12,485	12,866	12,127	12,065	12,299
	Calculated from ultimate analysis—									
	Calories.....								6,753	6,684
	B. t. u.....								12,155	12,391

DENVER No. 18.

Bituminous coal from Primero bed at Primero, Las Animas County, Colo., on the Colorado and Wyoming Railroad, was designated Denver No. 18.

One sample shipped from the bed consisted of 40 tons of run-of-mine coal and was used in making washing tests 243, 245, 246, and 248, and coking tests 262, 263, 264, 265, and 267.

Two mine samples were taken for chemical analysis. Sample 483-D was taken 4,200 feet west of the drift opening, where the coal measured as shown in section A; sample 484-D was taken 3,000 feet northwest of the drift opening, where the coal measured as shown in section B.

^a Not included in sample.

Section A (sample 483-D).			Section B (sample 484-D).		
	Ft.	in.		Ft.	in.
Coal ^a	5	½	Bone coal ^a	7	
Bone coal ^a	1	½	Coal.....	6	
Coal.....	1	4	Bone coal ^a	1	½
Bone coal ^a	1	½	Coal.....	10	
Coal.....	1	2	Bone coal.....		½
Bone coal.....	1		Coal.....	5	1
Coal.....	4	1		7	2
	7	4½			

Chemical analyses of Denver No. 18 coal.

	Mine samples.						Car sample (537-D).		
	483-D.			484-D.			Air dried.	As received.	Dry coal.
	Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.			
Air-drying loss.....	1.80	2.10	0.80
Prox. Moisture.....	.44	2.2343	2.5244	1.24
Volatile matter.....	32.66	32.07	32.80	33.27	32.58	33.42	31.91	31.65	32.05
Fixed carbon.....	57.97	56.93	58.23	55.49	54.32	55.73	51.40	50.99	51.63
(Ash.....	8.93	8.77	8.97	10.81	10.58	10.85	16.25	16.12	16.32
Ult. Sulphur.....	.58	.57	.58	.48	.47	.48	.54	.54	.55
Hydrogen.....	4.73	4.65
Carbon.....	69.96	70.88
Nitrogen.....	1.36	1.38
Oxygen.....	7.29	6.27
Calorific value:									
Determined—									
Calories.....	7,609	7,449	7,642	7,058	7,001	7,099
B. t. u.....	13,408	13,756	12,704	12,602	12,760
Calculated from ultimate analysis—									
Calories.....	6,981	7,099
B. t. u.....	12,566	12,724

DENVER No. 19.

A shipment of bituminous coal from the Nos. 4 and 5 beds at Taylor, King County, Wash., on the Columbia and Puget Sound Railroad, consisted of three grades of coal, designated Denver No. 19 A, B, and C. Denver No. 19A consisted of 25 tons of run-of-mine coal from the No. 5 bed and was used in making washing tests 250, 251, and 252, and coking tests 272, 273, and 294.

Denver No. 19B consisted of 15 tons of washed coal from the No. 5 bed and was used in making coking tests 269 and 270. Denver No. 19C consisted of 10 tons of run-of-mine coal from the No. 4 bed and was used in making washing test 253 and coking test 274.

Three mine samples were taken for chemical analysis. Sample 520-D was taken from No. 5 bed, 3,000 feet northwest of the drift mouth, where the coal measured as shown in section A; sample 518-D was taken from the same bed, 2,400 feet northeast of the drift mouth, where the coal measured as shown in section B; sample 519-D was taken from the No. 4 bed, 1,500 feet northeast of the drift mouth, where the coal measured as shown in section C.

^a Not included in sample.

Section A (sample 520-D).			Section B (sample 518-D).		
	Ft.	in.		Ft.	in.
Coal.....		5	Coal.....		5½
Shale and sandstone.....		½	Shale ^a		1
Coal.....		5½	Coal.....	1	1
Shale and sandstone ^a		1	Rash ^a		1½
Coal.....		1½	Coal.....	2	1
Rash ^a		2½	Foot wall, bone coal.....		
Coal.....	1	11		3	10
Foot wall, bone coal.....					
	3	2½			
			Section C (sample 519-D).		
			Coal.....	2	3½
			Bone coal.....		4
			Foot wall, bone coal.....		
				2	7½

Chemical analyses of Denver No. 19 coal.

		Mine samples.						Car sample 19A (586-D).		
		518-D.			519-D.					
		Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.
Prox.	Air-drying loss.....	2.20	3.30	2.30
	Moisture.....	1.92	4.08	2.13	5.36	3.12	5.35
	Volatile matter.....	36.00	35.88	37.41	38.97	37.68	39.81	34.97	34.17	36.10
	Fixed carbon.....	44.52	43.54	45.39	50.00	48.35	51.09	38.60	37.81	39.95
	Ash.....	16.87	16.50	17.20	8.90	8.61	9.10	23.22	22.67	23.95
Ult.	Sulphur.....	.55	.54	.55	.65	.63	.67	.79	.77	.81
	Hydrogen.....	4.72	4.36
	Carbon.....	56.98	60.20
	Nitrogen.....	1.37	1.45
	Oxygen.....	13.49	9.23
Calorific value:										
Determined—										
Calories.....		6,564	6,420	6,603	7,093	6,850	7,247	5,862	5,728	6,062
B. t. u.....			11,556	12,047		12,344	13,045	10,552	10,310	10,804
Calculated from ultimate analysis—										
Calories.....									5,665	5,965
B. t. u.....									10,197	10,773

		Mine sample (520-D).			Car sample 19C (585-D).		
		Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.
Prox.	Air-drying loss.....	3.50	2.60
	Moisture.....	2.76	6.16	3.70	6.20
	Volatile matter.....	41.43	39.98	42.61	35.11	34.20	36.46
	Fixed carbon.....	39.15	37.78	40.25	42.46	41.37	44.10
	Ash.....	16.66	16.08	17.14	18.73	18.23	19.44
Ult.	Sulphur.....	1.05	1.01	1.08	.71	.69	.74
	Hydrogen.....	4.95	4.54
	Carbon.....	60.12	64.09
	Nitrogen.....	1.41	1.50
	Oxygen.....	14.60	9.69
Calorific value:							
Determined—							
Calories.....					6,209	6,049	6,448
B. t. u.....					11,176	10,888	11,606
Calculated from ultimate analysis—							
Calories.....						5,948	6,241
B. t. u.....						10,706	11,414

^a Not included in sample.

DENVER No. 20.

Bituminous coal from the No. 3 bed at Carbonado, Pierce County, Wash., on the Northern Pacific Railroad, was designated Denver No. 20.

This shipment consisted of 50 tons of run-of-mine coal and was used in making washing tests 264, 265, and 266, and coking tests 287, 288, 289, 290, and 294.

One sample only, 522-D, was taken for chemical analysis. It was taken from the face of No. 3, north entry, 14,000 feet east of the drift mouth, where the coal measured as shown in section A.

Section A (sample 552-D).

	Ft.	in.
Coal.....	1	10
Shale ^a		4
Coal.....	1	10
Shale ^a		1½
Coal.....		10
Shale ^a		1
Coal.....		7
Shale ^a		2
Coal.....	2	4
Foot wall, bone coal.		
	8	1½

Chemical analyses of Denver No. 20 coal.

		Mine sample (552-D).			Car sample (787-D).		
		Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.
Prox.	Air-drying loss.....	1.80			3.30		
	Moisture.....	1.12	2.90		1.41	4.66	
	Volatile matter.....	31.51	30.94	31.87	30.06	29.07	30.49
	Fixed carbon.....	51.04	50.12	51.61	52.03	50.31	52.77
	Ash.....	16.33	16.04	16.52	16.50	15.96	16.74
Ult.	Sulphur.....	.47	.46	.47	.47	.45	.47
	Hydrogen.....					4.95	4.65
	Carbon.....					67.18	70.47
	Nitrogen.....					2.11	2.21
	Oxygen.....					9.35	5.46
Calorific value:							
Determined—							
Calories.....		7,037	6,907	7,117	6,970	6,740	7,070
B. t. u.....			12,433	12,810	12,546	12,132	12,726
Calculated from ultimate analysis—							
Calories.....						6,740	7,069
B. t. u.....						12,132	12,725

DENVER No. 21.

A shipment of bituminous coal from the Roslyn bed, 3 miles west of Roslyn, Kittitas County, Wash., on the Northern Pacific Railroad, consisted of two grades of coal, which were designated Denver

^a Not included in sample.

No. 21 A and B. Denver No. 21A consisted of 16 tons of run-of-mine coal and was used in making washing test 256 and coking tests 277 and 279. Denver No. 21B consisted of 34 tons of $\frac{3}{4}$ -inch screenings and was used in making washing tests 257, 258, and 270, and coking tests 278, 280, and 281.

Two mine samples were taken for chemical analysis. Sample 551-D was taken 1,300 feet southwest of the slope, where the coal measured as shown in section A; sample 550-D was taken 1,000 feet southwest of the slope, where the coal measured as shown in section B.

Section A (sample 551-D).			Section B (sample 550-D).		
	Ft.	in.		Ft.	in.
Coal.....	2	9 $\frac{1}{2}$	Coal.....	1	
Hard shale <i>a</i>		$\frac{1}{2}$	Mother coal.....		$\frac{1}{2}$
Coal.....	1	10	Coal.....	2	3
Shale <i>a</i>		$\frac{1}{2}$	Shale <i>a</i>		1
Coal.....		1 $\frac{1}{2}$	Coal.....		4
Floor, shale.			Shale <i>a</i>		$\frac{1}{2}$
	4	10 $\frac{1}{2}$	Coal.....		6 $\frac{1}{2}$
			Shale <i>a</i>		$\frac{1}{2}$
			Coal.....	1	1 $\frac{1}{2}$
			Shale <i>a</i>		$\frac{1}{2}$
			Coal.....		2
			Floor, shale.		
				4	9

Chemical analyses of Denver No. 21 coal.

	Mine samples.						Car samples.				
	550-D.			551-D.			21A (663-D).			21B (664-D).	
	Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.	As received.	Dry coal.
Air-drying loss.....	1.90			7.60			5.80				
Moisture.....	1.59	3.46		1.50	8.99		2.16	7.82		2.32	
Volatile matter.....	39.17	38.42	39.82	39.21	36.23	39.81	35.10	33.06	35.89	37.02	37.90
Fixed carbon.....	49.23	48.30	50.01	49.83	46.04	50.59	47.69	44.94	48.73	47.06	48.20
Ash.....	10.01	9.82	10.17	9.46	8.74	9.60	15.05	14.18	15.38	13.58	13.90
Sulphur.....	.42	.41	.43	.38	.35	.39	.48	.45	.49	.60	.61
Hydrogen.....								5.48	5.00	4.67	4.51
Carbon.....								64.67	70.16	66.77	68.36
Nitrogen.....								1.41	1.53	.61	.62
Oxygen.....								13.81	7.44	13.77	12.00
Caloric value:											
Determined—											
Calories.....	7,489	7,347	7,610				7,019	6,613	7,174	6,913	7,078
B. t. u.....		13,225	13,698				12,634	11,903	12,913	12,443	12,740
Calculated from ultimate analyses—											
Calories.....								6,829	7,083	6,425	6,578
B. t. u.....								11,751	12,749	11,565	11,840

a Not included in sample.

DENVER No. 22.

Bituminous coal from a mine 1 mile west of Lombard, Broadwater County, Mont., on the Northern Pacific Railroad, was designated Denver No. 22.

This shipment consisted of 45 tons of run-of-mine coal and was used in making washing tests 254 and 255 and coking tests 275 and 276.

Two mine samples were taken for chemical analysis. Sample 563-D was taken 300 feet north of the slope, where the coal measured as shown in section A; sample 564-D was taken 400 feet northeast of the slope, where the coal measured as shown in section B.

Section A (sample 563-D).

	Ft.	in.
Roof, coal.		
Coal.....	11	$\frac{1}{2}$
Shale and mother coal.....	$\frac{3}{4}$	
Coal.....	7	6
Floor, coal.	8	$8\frac{1}{4}$

Section B (sample 564-D).

	Ft.	in.
Roof, coal.		
Coal.....	1	3
Shale.....		$\frac{3}{4}$
Coal.....	5	3
Floor, coal.	6	$6\frac{1}{4}$

NOTE.—Small sulphur lenses are scattered promiscuously through the coal bed.

Chemical analyses of Denver No. 22 coal.

		Mine samples.						Car sample (662-D).		
		563-D.			564-D.			Air dried.	As received.	Dry coal.
		Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.			
Air-drying loss.....		2.30	3.10	1.60
Moisture.....		.49	2.7858	3.67	1.46	3.04
Prox. { Volatile matter.....		25.11	24.53	25.23	26.57	25.74	26.72	20.01	19.69	20.31
Fixed carbon.....		43.96	42.95	44.18	53.68	52.01	53.99	40.14	39.50	40.72
{ Ash.....		30.44	29.74	30.59	19.17	18.58	19.29	38.39	37.77	38.97
{ Sulphur.....		8.42	8.23	8.46	7.20	6.98	7.25	7.81	7.69	7.93
Ult. { Hydrogen.....		3.16	2.91
{ Carbon.....		44.16	45.54
{ Nitrogen.....	49	.49
{ Oxygen.....		6.73	4.16
Calorific value:										
Determined—										
Calories.....		5,722	5,590	5,750	4,576	4,503	4,643
B. t. u.....		10,062	10,350	8,237	8,105	8,357
Calculated from ultimate analysis—										
Calories.....		4,541	4,683
B. t. u.....		8,174	8,430

DENVER No. 23.

Bituminous coal from the Cherokee bed, 3 miles north of Frontenac, Crawford County, Kans., on the Atchison, Topeka and Santa Fe Railroad, was designated Denver No. 23.

This shipment consisted of 50 tons of run-of-mine coal and was used in making washing tests 259, 260, 261, 262, and 263, and coking tests 282, 283, 284, 285, and 286.

Two mine samples were taken for chemical analysis. Sample 660-D was taken 4,000 feet west of the shaft, where the coal measured as shown in section A; sample 661-D was taken 3,000 feet south of the shaft, where the coal measured as shown in section B.

Section A (sample 660-D).			Section B (sample 661-D).		
Roof, shale.	Ft.	in.	Roof, shale.	Ft.	in.
Coal.....	8		Coal.....	5½	
Shale and sandstone ^a	½		Sulphur.....	½	
Coal.....	2		Coal.....	2½	
Mother coal.....	½		Mother coal.....	½	
Coal.....	9½		Coal.....	10½	
Bone coal.....	½		Mother coal.....	½	
Coal.....	5½		Coal.....	1 6	
Mother coal and shale.....	½		Floor, fire clay.	3	½
Coal.....	9½				
Floor, fire clay.		2 11½			

Chemical analyses of Denver No. 23 coal.

		Mine samples.						Car sample (701-D).		
		660-D.			661-D.			Air dried.	As received.	Dry coal.
		Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.			
Prox.	Air-drying loss.....	3.30			4.10			2.80		
	Moisture.....	2.05	5.28		2.08	6.09		3.03	5.75	
	Volatile matter.....	35.11	33.95	35.85	34.20	32.80	34.93	34.24	33.28	35.28
	Fixed carbon.....	53.37	51.61	54.48	52.58	50.43	53.69	48.65	47.29	50.21
	Ash.....	9.47	9.16	9.67	11.14	10.68	11.38	14.08	13.08	14.51
Ult.	Sulphur.....	4.13	3.99	4.21	5.99	5.74	6.11	5.22	5.07	5.38
	Hydrogen.....								5.01	4.64
	Carbon.....								66.68	70.75
	Nitrogen.....								1.15	1.22
	Oxygen.....								8.41	3.50
Calorific value:										
Determined—										
Calories.....		7,476	7,229	7,632				6,915	6,721	7,132
B. t. u.			13,012	13,738				12,447	12,099	12,638
Calculated from ultimate analysis—										
Calories.....									6,866	7,285
B. t. u.									12,359	13,113

^a Not included in sample.

DENVER No. 24.

A shipment consisting of 25 tons of refuse from a washery at Starkville, Las Animas County, Colo., was designated Denver No. 24. Washing tests 276 and 277 were made. No mine samples were taken in connection with this shipment. No ultimate analysis of the coal was made.

Proximate chemical analysis of Denver No. 24 coal (car sample 838-D).

	As received.	Dry coal.
Moisture.....	2.64
Volatle matter.....	19.60	20.13
Fixed carbon.....	25.27	25.96
Ash.....	52.49	53.91
Sulphur.....	.41	.42

DENVER No. 25.

Bituminous coal from a mine working the "Cameron" bed, 1½ miles southwest of Rugby, Las Animas County, Colo., on the Colorado and Southern Railway, was designated Denver No. 25.

This shipment consisted of 40 tons of 1-inch screenings and was used in making washing tests 267, 268, and 269, and coking tests 291, 292, 295, and 296.

Two mine samples were taken for chemical analysis. Sample 734-D was taken 250 feet north of the drift mouth, where the coal measured as shown in section A; sample 735-D was taken 250 feet south of the drift mouth, where the coal measured as shown in section B.

Section A (sample 734-D).

	Ft.	in.
Roof, shale.		
Coal.....	1½	
Mother coal.....	¾	
Coal.....	6½	
Bone coal ^a	1½	
Coal.....	9½	
Sulphur.....	½	
Coal.....	7	
Bone coal.....	1½	
Coal.....	1	4½
Floor, shale.		
	3	8½

Section B (sample 735-D).

	Ft.	in.
Roof, shale.		
Coal.....	6½	
Bone coal ^a	1½	
Coal.....	5½	
Bone coal ^a	1½	
Coal.....	3½	
Sulphur.....	½	
Coal.....	1	11½
Floor, shale.		
	3	6½

^a Not included in sample.

Chemical analyses of Denver No. 25 coal.

		Mine samples.						Car sample (805-D).		
		734-D.			735-D.					
		Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.
Air-drying loss.....		1.10			1.60			1.40		
Prox.	Moisture.....	2.20	3.28		2.21	3.77		1.73	3.11	
	Volatile matter.....	36.48	36.08	37.31	37.14	36.55	37.98	35.72	35.22	36.35
	Fixed carbon.....	52.02	51.44	53.18	52.09	51.26	53.27	48.36	47.68	49.21
	Ash.....	9.30	9.20	9.51	8.56	8.42	8.75	14.19	13.99	14.44
Ult.	Sulphur.....	.73	.72	.74	.71	.70	.73	.82	.81	.84
	Hydrogen.....								4.86	4.65
	Carbon.....								67.60	69.77
	Nitrogen.....								1.36	1.40
	Oxygen.....								11.38	8.90
	Calorific value:									
Determined—										
Calories.....		7,258	7,178	7,421				6,870	6,775	6,992
B. t. u.			12,920	13,358				12,366	12,195	12,586
Calculated from ultimate analysis—										
Calories.....									6,666	6,880
B. t. u.									11,998	12,383

DENVER No. 26.

Subbituminous coal from the lower bed at Lafayette, Boulder County, Colo., on the Colorado and Southern Railway, was designated Denver No. 26.

This shipment consisted of 40 tons of run-of-mine coal and was used in making coking test 293.

Two mine samples were taken for chemical analysis. Sample 792-D was taken 1,500 feet southwest of the shaft, where the coal measured as shown in section A; sample 793-D was taken 3,000 feet west of the shaft, where the coal measured as shown in section B.

Section A (sample 792-D).

Roof, coal.	Ft. in.
Coal	2 11½
Mother coal and sulphur.....	½
Coal.....	7½
Mother coal and sulphur.....	½
Coal.....	3 5
Floor, bastard fire clay.	7 ½

Section B (sample 793-D).

Roof, coal.	Ft. in.
Coal	4 2
Bone coal.....	1½
Coal.....	2 9
Floor, bastard fire clay.	7 ½

Chemical analyses of Denver No. 26 coal.

		Mine samples.						Car sample (803-D).		
		792-D.			793-D.					
		Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.
Prox.	Air-drying loss.....	1.30			5.40			5.50		
	Moisture.....	17.70	18.77		10.58	21.08		14.58	19.28	
	Volatile matter.....	37.77	37.28	45.90	41.60	39.35	49.86	36.62	34.61	42.87
	Fixed carbon.....	40.50	39.97	49.20	38.22	36.16	45.82	43.83	41.41	51.30
	Ash.....	4.03	3.98	4.90	3.60	3.41	4.32	4.97	4.70	5.83
Ult.	Sulphur.....	.30	.30	.37	.36	.36	.46	.41	.39	.48
	Hydrogen.....								5.99	4.77
	Carbon.....								57.94	71.78
	Nitrogen.....								1.28	1.68
	Oxygen.....								29.70	15.56
Caloric value:										
Determined—										
Calories.....		5,678	5,004	6,899				5,917	5,591	6,926
B. t. u.....			10,067	12,418				10,661	9,064	12,467
Calculated from ultimate analysis—										
Calories.....									5,476	6,784
B. t. u.....									9,867	12,211

DENVER No. 27.

Bituminous coal from the "Berwind" bed, 1 mile west of Berwind, Las Animas County, Colo., on the Colorado and Southern Railway, was designated Denver No. 27.

This shipment consisted of 40 tons of run-of-mine coal and was used in making washing test 271 and coking tests 297, 300, 301, and 302.

As this was an undeveloped mine, only one mine sample was taken for chemical analysis. Sample 796-D was taken 180 feet southwest of the shaft, where the coal measured as shown in section A.

Section A (sample 796-D).

Roof, bone coal.	Ft. in.
Coal.....	5 11½
Floor, sandstone.	

Chemical analyses of Denver No. 27 coal.

		Mine sample (796-D).			Car sample (880-D).		
		Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.
Prox.	Air-drying loss.....	5.80			2.60		
	Moisture.....	.67	6.43		1.66	4.22	
	Volatile matter.....	33.78	31.82	34.00	33.24	32.38	33.80
	Fixed carbon.....	50.92	47.97	51.27	51.57	50.22	52.44
	Ash.....	14.63	13.78	14.73	13.53	13.18	13.76
Ult.	Sulphur.....	.59	.56	.60	.66	.64	.67
	Hydrogen.....					5.14	4.88
	Carbon.....					70.69	73.80
	Nitrogen.....					1.20	1.25
	Oxygen.....					9.15	5.64
Caloric value:							
Determined—							
Calories.....		7,207	6,789	7,256	7,233	7,045	7,355
B. t. u.....			12,220	13,061	13,019	12,681	13,239
Calculated from ultimate analysis—							
Calories.....						7,105	7,418
B. t. u.....						12,789	13,352

DENVER No. 28.

Bituminous coal from a bed at Cameo, Mesa County, Colo., on the Denver and Rio Grande and Colorado Midland railroads, was designated Denver No. 28.

This shipment consisted of 40 tons of 4-inch run-of-mine coal and was used in making washing test 272 and coking tests 298, 303, and 305.

Two mine samples were taken for chemical analysis. Sample 839-D was taken 3,500 feet northwest of the drift mouth, where the coal measured as shown in section A; sample 840-D was taken 3,700 feet northwest of the drift mouth, where the coal measured as shown in section B.

Section A (sample 839-D).			Section B (sample 840-D).		
	Ft.	in.		Ft.	in.
Roof, shale.			Roof, shale.		
Coal.....	1	10½	Coal.....	2	2½
Hard shale ^a		1	Shale ^a		¾
Coal.....	3	6	Coal.....	2	6
Floor, bone coal.			Shale and bone ^a		5
	5	5½	Coal.....	1	2½
			Floor, bone coal.		
				6	5

Chemical analyses of Denver No. 28 coal.

	Mine samples.						Car sample (852-D).		
	839-D.			840-D.			Air dried.	As received.	Dry coal.
	Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.			
Air-drying loss.....	1.70			2.10			3.60		
Moisture.....	5.50	7.11		5.44	7.43		5.10	8.52	
Volatile matter.....	35.82	35.21	37.90	36.98	36.20	39.11	35.25	33.98	37.15
Fixed carbon.....	49.15	48.31	52.01	49.65	48.62	52.52	46.89	45.20	49.40
Ash.....	9.53	9.37	10.09	7.92	7.75	8.37	12.76	12.30	13.45
Sulphur.....	.59	.58	.62	.57	.56	.61	.60	.58	.63
Hydrogen.....								5.38	4.84
Carbon.....								64.50	70.51
Nitrogen.....								1.31	1.43
Oxygen.....								15.93	9.14
Calorific value:									
Determined—									
Calories.....				6,915	6,770	7,314	6,523	6,288	6,873
B. t. u.....					12,186	13,165	11,741	11,318	12,371
Calculated from ultimate analysis—									
Calories.....								6,393	6,968
B. t. u.....								11,807	12,579

^a Not included in sample.

DENVER No. 29.

Bituminous coal from the upper part of the coal-basin bed at Coalbasin, Pitkin County, Colo., on the Crystal River Railroad, was designated Denver No. 29.

This shipment consisted of 25 tons of run-of-mine coal and was used in making washing tests 274 and 275 and coking tests 299, 306, and 307.

Two mine samples were taken for chemical analysis. Sample 5255 was taken on the second level on the right of the slope, where the coal measured as shown in section A; sample 5346 was taken in No. 58 slant, where the coal measured as shown in section B.

Section A (sample 5255).			Section B (sample 5346).		
	Ft.	in.		Ft.	in.
Coal (soft).....	3	3	Coal.....	6	6
Coal.....	2	4			
Bone coal ^a	1	4			
	6	11			

Chemical analyses of Denver No. 29 coal.

		Mine samples.						Car sample (894-D).		
		5255.			5346.					
		Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.
Prox.	Air-drying loss.....	0.40			1.20			1.80		
	Moisture.....	.80	1.20		1.20	2.39		1.29	3.07	
	Volatile matter.....	22.48	22.39	22.66	19.68	19.44	19.92	23.09	22.67	23.39
	Fixed carbon.....	73.20	72.90	73.79	71.62	70.76	72.49	66.29	65.10	67.16
	Ash.....	3.52	3.51	3.55	7.50	7.41	7.59	9.33	9.16	9.45
	Sulphur.....	.74	.74	.75	.53	.52	.53	.64	.63	.65
	Hydrogen.....								4.96	4.77
	Carbon.....								78.81	81.31
	Nitrogen.....								1.69	1.74
	Oxygen.....								4.75	2.08
Ult.	Calorific value:									
	Determined—									
	Calories.....				8,084	7,987	8,183	7,915	7,772	8,018
	B. t. u.....					14,377	14,729	14,247	13,990	14,432
	Calculated from ultimate analysis—									
	Calories.....								7,888	8,138
	B. t. u.....								14,198	14,648

DENVER No. 30.

Bituminous coal from the lower part of the coal-basin bed (not being worked at present) at Coalbasin, Pitkin County, Colo., on the Crystal River Railroad, was designated Denver No. 30.

This shipment consisted of 17 tons of run-of-mine coal and was used in making washing tests 273 and 275 and coking tests 304 and 307.

^a Not included in sample.

This coal is from the same bed as Denver No. 29, being the lower part of it. This is the floor coal of the sections shown in Denver No. 29, and is called bone coal because it contains more impurities than the upper part of the coal bed

Two mine samples were taken for chemical analysis. Sample 5262 was taken in the main slope air course, where the coal measured as shown in section A; sample 5249 was taken in the second level on the right of the slope, where the coal measured as shown in section B.

Section A (sample 5262).			Section B (sample 5249).		
Coal. ^a	Ft.	in.	Coal ^a	Ft.	in.
Bone coal.....	3	10	Bone coal.....	5	0
				11	6

Chemical analyses of Denver No. 30 coal.

		Mine samples.						Car sample (888-D).		
		5262.			5249.					
		Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.
Prox.	Air-drying loss.....	0.40	1.67	0.30	1.60	3.10
	Moisture.....	1.28	1.30	1.60	3.75
	Volatile matter.....	21.41	21.32	21.68	19.66	19.60	19.92	22.83	22.12	22.98
	Fixed carbon.....	67.99	67.73	68.88	59.56	59.38	60.34	64.40	62.41	64.84
	Ash.....	9.32	9.28	9.44	19.48	19.42	19.74	12.10	11.72	12.18
Ult.	Sulphur.....	.47	.47	.48	.45	.45	.46	.41	.40	.42
	Hydrogen.....	4.79	4.79	4.54
	Carbon.....	75.80	75.80	78.75
	Nitrogen.....	1.45	1.45	1.61
	Oxygen.....	5.84	5.84	2.60
Calorific value:										
Determined—										
Calories.....		7,843	7,812	7,945	7,653	7,416	7,704
B. t. u.....		14,062	14,301	13,755	13,349	13,867
Calculated from ultimate analysis:										
Calories.....		7,533	7,826
B. t. u.....		13,559	14,067

DENVER No. 31.

Bituminous coal from the "Carthage" bed, at Carthage, Socorro County, N. Mex., on the New Mexico Midland Railroad, was designated Denver No. 31.

This shipment consisted of 30 tons of run-of-mine coal and was used in making washing test 279 and coking tests 311, 312, and 313.

Two mine samples were taken for chemical analysis. Sample 890-D was taken 870 feet south of the slope mouth, where the coal measured as shown in section A; sample 889-D was taken 700 feet south of the slope mouth, where the coal measured as shown in section B.

^a Not included in sample.

Section A (sample 890-D).

Section B (sample 889-D).

	Ft.	in.		Ft.	in.
Roof, bone coal.			Roof, sandstone.		
Coal.....	1	$\frac{1}{2}$	Coal.....	1	2
Shale ^a		$1\frac{1}{4}$	Shale ^a		6
Coal.....	1	2	Coal.....		$6\frac{1}{2}$
Shale and sulphur ^a		$\frac{3}{4}$	Sulphur.....		$\frac{1}{2}$
Coal.....	1	$\frac{3}{4}$	Coal.....	1	$7\frac{1}{2}$
Bastard fire clay ^a	2		Bastard fire clay ^a		3
Coal.....		$8\frac{1}{2}$	Coal.....		$1\ 6\frac{1}{2}$
Coal and shale.....		$\frac{3}{4}$	Floor, shale.		
Coal.....		$8\frac{1}{2}$		5	$7\frac{1}{2}$
Floor, shale.					
	5	$4\frac{1}{2}$			

Chemical analyses of Denver No. 31 coal.

	Mine samples.						Car sample (972-D).		
	889-D.			890-D.					
	Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.
Air-drying loss.....	1.30			2.10			0.60		
Moisture.....	2.08	3.35		1.85	3.91		2.37	2.96	
Volatile matter.....	39.96	39.44	40.80	39.70	38.87	40.44	37.43	37.20	38.33
Fixed carbon.....	50.50	49.85	51.58	47.83	46.82	48.74	45.39	45.12	46.50
Ash.....	7.46	7.36	7.62	10.62	10.40	10.82	14.81	14.72	15.17
Sulphur.....	.84	.83	.86	.72	.70	.73	.80	.79	.81
Hydrogen.....								4.78	4.59
Carbon.....								67.73	69.80
Nitrogen.....								1.26	1.30
Oxygen.....								10.71	8.32
Caloric value:									
Determined—									
Calories.....				7,231	7,079	7,367	6,827	6,786	6,993
B. t. u.....					12,742	13,261	12,289	12,215	12,587
Calculated from ultimate analysis—									
Calories.....								6,676	6,880
B. t. u.....								12,017	12,384

DENVER No. 32.

Bituminous coal from the Yampa or Lower bed, at Oak Creek, Routt County, Colo., on the Denver, Northwestern and Pacific Railroad, was designated Denver No. 32.

This shipment consisted of 40 tons of run-of-mine coal and was used in making washing test 278 and coking tests 308, 309, and 310.

Two mine samples were taken for chemical analysis. Sample 916-D was taken from the face of the main slope, 725 feet northwest of the slope mouth, where the coal measured as shown in section A; sample 915-D was taken from the face of the south slope, 700 feet northwest of the mouth, where the coal measured as shown in section B.

^a Not included in sample.

<i>Section A (sample 916-D).</i>			<i>Section B (sample 915-D).</i>		
Roof, sandstone.	Ft.	in.	Roof, sandstone.	Ft.	in.
Coal.....		6½	Coal.....		6½
Shale.....		½	Shale.....		½
Coal.....	1	6½	Coal.....	1	7½
Hard shale ^a		10	Hard shale ^a		10
Coal.....	2	7½	Coal.....	2	9½
Floor, shale.			Floor, shale.		
	5	6½		5	10½

Chemical analyses of Denver No. 32 coal.

	Mine samples.						Car sample (971-D).		
	915-D.			916-D.					
	Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.
Air-drying loss.....							0.90		
Moisture.....		6.43			6.16		6.69	7.53	
Volatile matter.....		37.36	39.93		38.12	40.62	36.68	36.36	39.32
Fixed carbon.....		47.58	50.85		47.92	51.07	45.88	45.46	49.16
Ash.....		8.63	9.22		7.80	8.31	10.75	10.65	11.52
Sulphur.....		1.53	1.63		1.10	1.17	1.52	1.51	1.63
Hydrogen.....								5.29	4.81
Carbon.....								64.90	70.19
Nitrogen.....								1.41	1.52
Oxygen.....								16.24	10.33
Calorific value:									
Determined—									
Calories.....					6,707	7,147	6,423	6,366	6,884
B. t. u.....					12,073	12,865	11,561	11,459	12,391
Calculated from ultimate analysis—									
Calories.....								6,400	6,921
B. t. u.....								11,520	12,456

DENVER No. 33.

Bituminous coal from the No. 7 bed at Sweetwater, Sweetwater County, Wyo., on the Union Pacific Railroad, was designated Denver No. 33.

This shipment consisted of 40 tons of run-of-mine coal and was used in making washing test 280 and coking tests 314 and 315.

Two mine samples were taken for chemical analysis. Sample 946-D was taken approximately 1½ miles north of the drift, where the coal measured as shown in section A; sample 945-D was taken 1½ miles north of the drift, where the coal measured as shown in section B.

<i>Section A (sample 946-D).</i>			<i>Section B (sample 945-D).</i>		
Roof, sandy shale.	Ft.	in.	Roof, sandy shale.	Ft.	in.
Coal.....	1	11	Coal.....	1	11
Shale.....		½	Shale.....		½
Coal.....	3	11½	Coal.....	4	½
Floor, sandstone.			Floor, sandstone.		
	5	11		6	00

^a Not included in sample.

Chemical analyses of Denver No. 33 coal.

	Mine samples.						Car sample (973-D).		
	945-D.			946-D.					
	Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.	Air dried.	As received.	Dry coal.
Air-drying loss.....	1.40	1.4040
Moisture.....	7.51	8.80	7.74	9.03	8.27	8.64
Volatile matter.....	39.44	38.90	42.65	38.98	38.43	42.24	38.62	38.47	42.11
Fixed carbon.....	50.16	49.46	54.24	50.61	49.91	54.87	48.50	48.30	52.87
Ash.....	2.88	2.84	3.11	2.67	2.63	2.89	4.61	4.59	5.02
Sulphur.....	1.00	.99	1.00	.89	.88	.97	1.09	1.09	1.19
Hydrogen.....	5.53	5.00
Carbon.....	69.55	76.13
Nitrogen.....	1.53	1.67
Oxygen.....	17.71	10.99
Caloric value:
Determined—
Calories.....	7,033	6,935	7,623	6,791	6,764	7,404
B. t. u.....	12,483	13,721	12,224	12,175	13,327
Calculated from ultimate analysis—
Calories.....	6,788	7,430
B. t. u.....	12,218	13,373

WASHING TESTS.

By G. R. DELAMATER.

INTRODUCTORY STATEMENT.

The equipment and operation of the Denver washery in the previous tests were described in Bulletin 368. In brief, the run-of-mine coal from the railroad car was shoveled to a hopper scale, whence it passed through a toothed-roll crusher that broke it down to a maximum of 2½ inches. Thence an apron conveyor and a bucket elevator took the raw coal, which was sampled at the elevator, to one of six storage bins. The sample of coal was analyzed and float-and-sink tests were made on it to determine the size giving the best separation. From the storage bin the coal went to a corrugated-roll disintegrator, where it was crushed to the desired size; thence another elevator carried it to a storage bin. From this it went to the sluice boxes leading to the jigs. A Richards jig and a special jig were used. The washed coal was finely crushed by a Pennsylvania hammer crusher, then elevated to the top of the washery building and conveyed to a chute supplying the coke-oven larry.

CHANGES IN EQUIPMENT.

For the tests described in this bulletin a few changes were made in the equipment.

One 50-barrel water tank was installed and the water lines from the pump were changed to deliver direct to this tank, thus giving a head of about 30 feet to the water for the jigs. This arrangement gave much better results, as will be noted, for instance, in the tests on

the Montana coals, in which much better separations were made than were made the year before on similar coals. Changes in the sludge-tank arrangement could not be made, and in consequence the loss in the refuse continued. Riffing the tank brought about some improvement in some of the coals tested, but not in all.

A Richards regulation 4-inch four-compartment pulsator ore jig was tried in some of the tests, and later a 4-inch six-compartment jig of the same type was tried with interesting results. In some of the tests the refuse from these jigs was remarkably free from coal, but in none was the ash reduction satisfactory. However, it must be remembered that neither jig was built for cleaning coal, but was a regular ore jig. The writer believes the pulsations are as nearly perfect as in any jig, and that with careful redesigning to suit the conditions peculiar to coal, a machine can be built which will greatly assist in overcoming some of the present difficulties of coal washing. The main trouble with the jig as now constructed appears to be that the jig takes off the heavier refuse in the first two or three compartments, and it is practically impossible to obtain a bed in the others. The machine now has no trouble in removing the heavier refuse matter, but is ineffective in separating the bone and refuse matter that has a specific gravity close to that of the good coal. This defect is probably due to the fact that the designer of the machine had ore practice only in mind, in which work the specific gravity of the tailings is decidedly less than that of the concentrates and the product of each compartment is different and is kept separate after leaving the machine. However, as previously stated, the writer believes that the machine can be designed to overcome the different conditions of coal washing, and those concerned in this subject will watch with interest the development of this machine for use in the washing of coal.

FLOAT-AND-SINK TESTS.

The float-and-sink tests were made with machines, and as large a sample as possible was used, usually 2 kilograms to a test. Whereas with the old system of skimming and filtering there was difficulty in making an average of two tests a day, with the new machines now in use it is easily possible to make 30 to 35 tests a day. Another great advantage is that the liquid is not removed from the tanks throughout the tests. Much time and labor is thus saved, though, of course, most of the extra time consumed in the old method was taken up by the filtering.

Figure 1 shows the float-and-sink machine. It is composed of a wooden box about 30 inches long and 12 inches wide, with double ends and partitions *m, m* dividing the box into three compartments. The central compartment contains the specific gravity solution and is lined throughout with sheet zinc. Two standard laboratory sieves, *g*, with

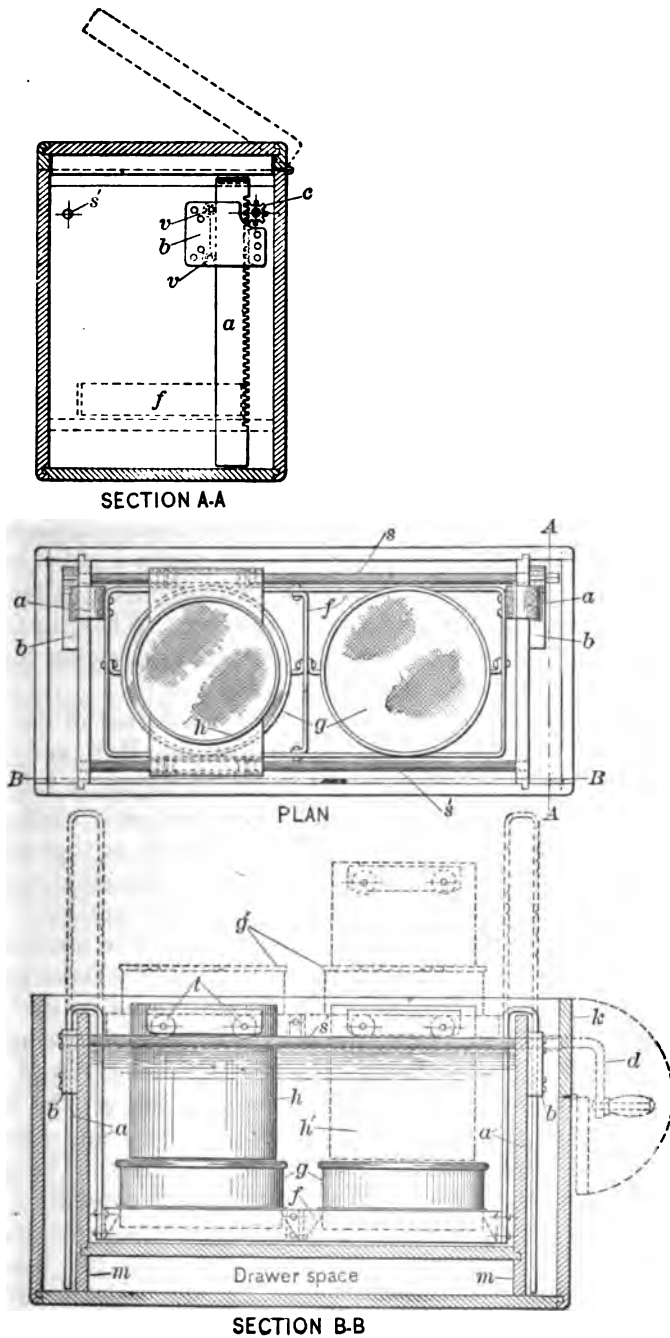


FIGURE 1.—Float-and-sink test machine.

60-mesh brass cloth bottoms, are supported by a light iron frame, *f*. On the rear corners at each end of this frame *f* two pieces of strap iron, *a, a*, bent in the form of a U are riveted in the manner shown in section B-B. They straddle the partitions *m, m*; the rear edges of the outside legs have teeth cut therein, as shown in section A-A, and mesh with pinions, *c*, which are supported on a shaft, *s*. Guides, *b, b*, hold the frame extensions *a, a* in position, the rollers *v, v* allowing free vertical movement of the frame extensions *a, a*. When the box lid is thrown back the upper portion *k* of the box end may be opened as shown, the handle *d* then being used in turning the shaft *s* and the frame *f* being raised or lowered as desired. The outside legs of the frame extensions *a, a* are the longer, so that they may still remain in the guides *b, b* when the frame *f* is in its highest position. For this reason a false bottom is used in the central compartment to reduce the amount of liquid necessary to fill the tank, and the space beneath the false bottom is used for drawers, where the machine accessories may be kept.

The cylinder *h* is of thin plate, the diameter being a little less than that of the sieve pans; it is supported on the shafts *s* and *s'* as shown, the wheels *t* resting directly on *s* and *s'* and permitting a free movement of the cylinder from end to end of the box. When the frame *f* is in its lowest position the bottom of the cylinder *h* clears the top of the sieve pans by about an eighth of an inch.

The central compartment is filled with a solution of the desired specific gravity to about the level shown in section B-B, and the sieve pans and cylinder are placed in the positions shown in full outline, section B-B. The frame *f* is elevated until the sieve cloth touches the bottom of the cylinder *h*, and the coal sample is then deposited inside the cylinder *h*. After proper stirring to free all the particles, those lighter than the liquid float and those heavier settle to the bottom of the cylinder, resting on the screen cloth. The machine is left in this position for a few moments; then frame *f* is lowered to the bottom. The cylinder *h* is then very slowly moved to the position shown by the dotted outline *h'*, the float in the cylinder being carried over with it. The frame *f* is then elevated very slowly until entirely out of the liquid, as shown in the dotted outline *g'*, the liquid in the sieve pans filtering through the cloth bottoms and leaving the sink in the left-hand pan and the float in the other. The pans are then lifted from the frame *f* and the samples are carefully rinsed in the pans with water by means of a spray, all particles adhering to the cylinder *h* being washed into the pan containing the float. The samples are then weighed in the pans, the pan weights subtracted therefrom, and the percentages of float and sink calculated. The float sample is then ready for analysis.

Only one part of the operation of this machine requires very careful manipulation. When the cylinder is moved to the position *g'*, it must be moved very slowly in order that there may be no disturbance of the liquid that would cause some of the float coal to be sucked down and out of the cylinder. This danger was clearly indicated in the work of this plant. After about 150 tests had been made on these machines, practically no pieces were found either floating on the liquid or in the bottom of the tanks; but a new operator lost a considerable amount in making only two tests, owing entirely to moving the cylinder over too rapidly and with a short, jerky movement. A better arrangement would be to move the cylinder by means of a small chain winding on a suitable drum. In this manner the movement of the cylinder would be uniform and as slow as desired. The pan frame should also be elevated very slowly, for otherwise, if the pans are nearly full, the pieces will be washed out.

DETAILS OF TESTS.

In the following table, in column headed "Loss of good coal in refuse," good coal means all free coal in the refuse the analysis of which is the same or nearly the same as that of the washed coal. The percentage of loss expresses the relation of the good coal in the refuse to the total of the good coal in the refuse and the washed coal. It is obtained by the following formula:

Let

x = the percentage of loss of good coal.

a = the percentage of washed coal in the sample.

b = the percentage of refuse in the sample.

c = the percentage of good coal (as defined above) in the refuse.

d = the percentage expressing the relation of the good coal in the refuse to the raw sample; that is, $bc = d$.

Then

$$x = \frac{100d}{a + d}$$

General data of washing tests at Denver, Colo., 1908-9.

Denver No.	Test No.	Date.	Duration.	Size of coal.		Jig used. ^a		Raw coal used.	Washed coal.		Refuse.		Loss of good coal in refuse.
				As shipped.	As washed.	Name.	Speed.		Amount.	Per cent.	Amount.	Per cent.	
		1908.	H. m.				R.p.m.	Tons.	Tons.		Tons.	P.ct.	
15A	232	Aug. 25	2 35	R. o. m.	$\frac{3}{4}$ inch.	Special.	112	7.36	5.50	75	1.86	25	18
15A	233	Aug. 27	2 45	do.	do.	do.	108	6.90	5.33	77	1.57	23	16
15A	234	Aug. 28	1 45	do.	do.	do.	108	7.13	6.27	88	.86	12	8
15A	235	Aug. 29	2	do.	do.	do.	108	6.97	6.43	92	.54	8	3
15B	236	Sept. 1	2 20	Black.	Black.	do.	108	6.57	5.32	81	1.25	19	10
15B	238	Sept. 4	5	do.	$\frac{3}{4}$ inch.	Richards.	86	5.91	5.59	95	.32	5	0.4
15B	239	Sept. 7	6	do.	do.	do.	86	6.51	6.11	94	.40	6	2
15B	240	Sept. 8	2 45	do.	do.	do.	86	6.01	5.81	97	.20	3	0.7
16	242	Sept. 14	3 55	R. o. m.	$\frac{3}{4}$ inch.	Special.	108	14.78	12.63	85	2.15	15	6
16	247	Sept. 23	3 30	do.	$\frac{3}{4}$ inch.	Richards.	86	13.24	11.23	85	2.11	15	11
16	249	Sept. 28	3 20	do.	do.	do.	86	11.36	9.66	85	1.70	15	11
17	237	Sept. 2	5 10	do.	$\frac{3}{4}$ inch.	Special.	108	18.14	12.88	71	5.26	29	19
17	241	Sept. 9	4 15	do.	do.	Richards.	86	4.69	3.99	85	.70	15	7
17	244	Sept. 16	2	do.	do.	do.	86	8.00	6.40	80	1.60	20	11
18	243	Sept. 15	3 10	do.	$\frac{3}{4}$ inch.	Special.	108	13.99	11.35	81	2.64	19	6
18	245	Sept. 19	1 35	do.	$\frac{3}{4}$ inch.	Richards.	86	5.63	4.90	87	.73	13	3
18	246	Sept. 21	1 50	do.	do.	do.	86	5.99	4.40	74	1.59	26	18
18	248	Sept. 24	1 40	do.	$\frac{3}{4}$ inch.	Special.	108	7.19	5.97	83	1.22	17	5
19A	250	Oct. 2	2	do.	$\frac{3}{4}$ inch.	Richards.	86	6.67	3.80	57	2.87	43	28
19A	251	Oct. 3	1 55	do.	$\frac{3}{4}$ inch.	Special.	110	8.66	5.80	67	2.86	33	17
19A	252	Oct. 6	3 15	do.	$\frac{3}{4}$ inch.	Richards.	86	7.06	4.65	66	2.41	34	21
19C	253	Oct. 7	3 40	do.	do.	do.	86	6.51	4.33	67	2.18	33	22
22	254	Oct. 13	6 30	do.	do.	do.	75	6.76	4.73	70	2.03	30	6
22	255	Oct. 15	5 30	do.	$\frac{3}{4}$ inch.	Special.	110	13.55	4.20	31	9.35	69	25
21A	256	Oct. 21	2	do.	do.	do.	110	8.92	7.83	88	1.09	12	3
21B	257	Oct. 21	55	$\frac{3}{4}$ inch.	do.	do.	120	7.43	6.30	85	1.13	15	8
21B	258	Oct. 23	4 25	$\frac{3}{4}$ inch.	do.	Richards.	75	6.25	5.01	80	1.24	20	14
21B	270	Nov. 28	3	do.	do.	do.	81	3.11	2.77	89	.34	11	8
23	259	Oct. 24	2 10	R. o. m.	$\frac{3}{4}$ inch.	Special.	120	6.48	4.34	67	2.14	33	18
23	260	Oct. 27	1 55	do.	do.	do.	108	10.58	6.24	59	4.34	41	20
23	261	Oct. 27	3	do.	do.	do.	108	10.09	6.90	68	3.19	32	20
23	262	Oct. 30	6	do.	$\frac{3}{4}$ inch.	Richards.	75	8.23	6.34	77	1.89	23	17
23	263	Nov. 3	2 15	do.	$\frac{3}{4}$ inch.	Special.	108	7.30	5.65	77	1.65	23	11
20	264	Nov. 5	4 40	do.	do.	do.	108	10.68	10.04	94	.64	6	1
20	265	Nov. 6	2 40	do.	do.	do.	108	9.01	6.88	76	2.13	24	6
20	266	Nov. 7	5 20	do.	do.	do.	108	11.84	10.06	86	1.78	15	5
25	267	Nov. 12	3 10	1-inch scr'gs.	$\frac{3}{4}$ inch.	do.	108	14.83	12.16	82	2.67	18	6
25	268	Nov. 17	40	do.	do.	do.	108	3.82	3.13	82	.69	18	11
25	269	Nov. 21	3	do.	do.	Richards.	96	4.53	3.94	87	.59	13	9
27	271	Dec. 4	3 20	R. o. m.	$\frac{3}{4}$ inch.	Special.	108	18.51	15.55	84	2.96	16	8
28	272	Dec. 5	1 25	do.	do.	do.	108	9.74	8.50	87	1.24	13	4
30	273	Dec. 7	2	do.	do.	do.	108	7.71	6.55	85	1.16	15	6
29	274	Dec. 9	1 25	do.	do.	do.	108	10.70	6.43	60	4.27	40	29
30	275	Dec. 10	2 30	do.	do.	do.	108	11.14	9.02	81	2.12	19	9
30	276	Dec. 14	4 30	$\frac{1}{2}$ inch.	$\frac{3}{4}$ inch.	Richards.	81	13.58	2.58	19	11.00	81	57
24	277	Dec. 1	5	do.	$\frac{1}{2}$ inch.	do.	81	8.89	1.60	18	7.29	82	44
32	278	Dec. 23	3 10	R. o. m.	$\frac{3}{4}$ inch.	Special.	108	13.69	12.00	88	1.69	12	4
31	279	Dec. 26	2 25	do.	do.	do.	108	12.86	11.10	86	1.76	14	2
33	280	Dec. 29	3	do.	do.	do.	108	10.90	10.68	98	.22	2	0.7

^a The special jig has a $\frac{3}{4}$ -inch stroke. In the Richards jig the number of pulsations per minute is twice the number of revolutions.

As in the previous year's work, all analyses are reduced to a dry basis to afford better comparison of the test results.

In the following tables of analyses the columns headed "Percentage of reductions" show the proportionate difference between the percentage of the stated impurity in the raw coal and in the washed coal. The columns headed "Percentage removed" give, with reference to the quantity of the stated impurity in the raw coal, the proportion removed by washing. These figures are determined by the following formulæ:

Let

X = the percentage of reduction of any constituent.

Y = the percentage of any constituent removed by washing.

M = the percentage that the amount of the constituent in the washed coal is of the raw coal.

a = the percentage that the washed coal is of the raw coal.

b = the percentage of the constituent in the washed coal.

c = the percentage of the constituent in the raw coal.

Then

$$X = \frac{(c-b)}{c}, M = ab, \text{ and } Y = \frac{(c-M)}{c}$$

Analyses of coals at Denver, Colo., 1908-9.

[All reduced to a dry basis for better comparison.]

Denver No.	Raw coal.				Washing test No.	Washed coal.												Refuse.			
	Volatile matter.	Fixed carbon.	Ash.	Sulphur.		Volatile matter.	Fixed carbon.	Ash.			Sulphur.			Volatile matter.	Fixed carbon.	Ash.	Sulphur.				
								Per cent.	Per cent of reduction.	Per cent re-moved.	Per cent.	Per cent of reduction.	Per cent re-moved.								
15A	37.51	53.10	9.39	1.23	232	38.68	54.09	7.23	23	42	1.12	9	32	35.07	49.64	15.29	1.92				
					233	38.45	54.67	6.88	27	37	1.08	12	24	31.69	50.51	17.80	1.73				
					234	40.60	52.19	7.21	23	32	1.08	12	23	32.12	45.01	22.87	1.83				
					235	39.25	53.23	7.52	20	26	1.10	11	18	26.81	40.40	32.79	2.61				
					236	39.53	53.23	7.24	42	48	1.09	11	7	28.86	36.36	34.78	1.91				
15B	36.30	51.22	12.48	1.04	238	37.10	52.59	10.31	17	22	1.14	11	8	17.32	9.38	73.30	2.04				
					239	37.55	53.72	8.73	30	34	1.02	12	8	24.70	19.74	55.56	2.34				
					240	35.31	54.60	10.09	19	22	.99	5	7	19.45	24.91	55.64	1.97				
16	31.43	54.07	14.50	.66	242	31.67	56.74	11.59	20	32	.67	2	14	22.42	36.42	41.16	.75				
					247	31.82	55.78	12.40	14	27	.66	1	15	25.61	45.04	29.35	.68				
					249	30.85	56.84	12.31	15	28	.73	1	6	26.82	47.80	25.38	.55				
17	31.96	49.88	18.16	.69	237	34.12	52.97	12.91	29	44	.74	1	16	27.56	41.18	31.26	1.03				
					241	35.22	50.09	14.69	19	22	.69	1	4	25.30	36.87	37.83	.79				
					244	33.23	51.80	14.97	18	26	.61	12	19	26.74	42.00	31.26	.69				
					243	32.89	57.41	9.70	41	52	.52	5	24	25.24	34.64	40.12	.50				
18	32.05	51.63	16.32	.55	245	31.63	56.26	12.11	26	32	.53	4	13	25.55	29.19	45.26	1.88				
					246	31.15	56.20	12.65	22	43	.59	2	20	28.43	43.12	28.45	.46				
					248	33.16	54.98	11.86	27	44	.52	5	27	26.81	34.36	38.83	.45				
19A	36.10	39.95	23.95	.81	250	35.72	46.98	17.30	28	44	.83	1	20	34.13	32.95	32.92	.84				
					251	37.03	46.84	16.13	31	55	.92	1	23	38.96	31.91	39.13	.99				
19C	36.46	44.10	19.44	.74	252	35.45	45.65	18.90	21	48	.76	6	38	33.60	32.39	34.01	1.04				
22	20.31	40.72	38.97	7.93	253	36.56	48.04	15.40	21	47	.70	5	36	32.52	39.50	27.98	.73				
21A	34.06	49.86	16.08	.82	254	24.10	45.64	30.26	22	55	4.17	47	69	17.61	23.33	59.06	11.16				
					255	27.65	57.76	14.59	62	84	3.39	57	81	19.34	30.48	50.18	13.79				
21B	37.90	48.20	13.90	.61	256	36.35	53.23	10.42	35	43	.74	10	21	24.68	26.34	48.98	2.21				
					257	38.42	51.11	10.47	25	36	.67	1	7	29.88	35.69	34.43	2.16				
					258	38.18	50.59	11.23	19	32	.62	1	15	34.00	41.12	24.88	.99				
					270	35.74	52.07	12.19	12	29	.52	15	31	30.18	41.60	28.22	.80				
					259	38.69	54.29	7.02	52	60	3.29	39	50	31.39	38.57	30.04	6.34				
23	35.28	50.21	14.51	5.38	260	38.44	55.33	6.23	57	70	3.10	42	58	32.39	41.12	26.49	6.78				
					261	37.65	55.89	6.46	55	70	3.43	36	57	29.05	47.11	33.84	8.51				
					262	37.33	52.95	9.72	33	58	4.15	23	51	28.82	40.94	30.24	10.40				
					263	37.85	54.22	7.93	45	58	3.86	28	45	25.75	36.64	37.61	9.24				
					264	32.43	53.21	14.36	14	34	.58	1	4	21.38	22.65	55.97	.71				
20	30.49	52.77	16.74	.47	265	31.36	54.84	13.80	18	37	.57	1	9	No sample.							
					266	32.68	55.52	11.80	30	55	.56	1	23	23.64	30.85	45.51	.42				
					267	38.37	52.04	9.59	34	48	.73	13	32	27.33	36.69	35.98	.55				
25	36.35	49.21	14.44	.84	268	38.49	50.75	10.76	25	47	.75	11	36	30.85	30.85	30.72	.69				
					269	36.32	51.83	11.85	18	38	.84	1	25	28.69	40.15	31.16	.94				
27	33.80	52.44	13.76	.67	271	33.71	55.61	10.68	22	38	.62	7	25	28.22	41.72	30.06	.81				
28	37.15	49.40	13.45	.63	272	36.24	54.65	9.11	32	41	.61	3	16	28.66	34.57	36.77	.85				
30	22.98	64.84	12.18	.42	273	24.55	66.14	9.31	24	35	.46	1	7	22.44	50.54	27.02	.52				
29	23.39	67.16	9.45	.65	274	23.34	70.17	6.49	31	40	.52	20	31	23.48	62.60	13.92	.94				
29	23.18	66.01	10.81	.53	275	23.79	68.23	7.98	26	37	.48	9	23	21.57	55.51	22.92	.74				
24	20.13	25.96	53.91	.42	276	23.79	42.10	34.11	37	81	.46	1	67	20.35	31.14	58.51	.53				
					277	25.34	40.59	34.07	37	84	.49	1	71	18.45	23.27	58.28	.74				
32	39.32	49.16	11.52	1.63	278	40.86	50.97	8.17	29	38	1.54	6	17	30.08	37.47	32.45	1.54				
31	38.33	46.50	15.17	.81	279	40.93	48.98	10.09	33	43	.82	1	12	24.96	25.86	49.18	1.48				
33	42.11	52.87	5.02	1.19	280	42.38	53.43	4.19	16	22	.90	24	29	27.87	31.57	40.56	3.26				

Raw-coal float-and-sink tests.

Denver No.	Test No.	Maximum size of coal.	Specific gravity of solution used.	Per-centage of—		Float-coal analyses (dry basis).		Denver No.	Test No.	Maximum size of coal.	Specific gravity of solution used.	Per-centage of—		Float-coal analyses (dry basis).	
				Float.	Sink.	Ash.	Sulphur.					Float.	Sink.	Ash.	Sulphur.
15A	1	2 inches	1.35	73	27	6.01	1.05	24	185	(a)	1.35	7	93	7.86	0.75
	2	do.	1.45	96	4	9.07	.92		186	do.	1.42	15	85	10.26	.62
	3	1 inch.	1.35	77	23	5.56	.93		187	(a)	1.46	18	82	12.39	.61
	4	do.	1.45	94	6	7.35	.91		188	(a)	1.55	25	75	14.72	.55
15B	5	Slack	1.36	85	15	5.64	.99	25	197	1 inch.	1.35	70	30	6.18	.69
	6	do.	1.45	87	13	6.19	1.01		198	do.	1.40	80	20	7.19	.68
17	7	1/2 inch.	1.35	48	52	7.20	.67		199	do.	1.46	85	15	8.14	.66
	8	do.	1.45	78	22	12.03	.71		200	do.	1.55	88	12	8.40	.64
16	9	do.	1.35	68	32	6.98	.68	24	201	1/2 inch.	1.35	8	92	8.78	.62
	10	do.	1.45	86	14	9.96	.66		202	do.	1.40	10	90	9.94	.50
	11	1/2 inch.	1.35	65	35	8.29	.74		203	do.	1.46	20	80	12.15	.63
	12	do.	1.45	84	16	11.62	.66		204	do.	1.55	24	76	17.16	.60
18	13	do.	1.35	73	27	8.03	.58	28	221	do.	1.35	68	32	6.21	.60
	14	do.	1.45	81	19	10.00	.54		222	do.	1.40	85	15	7.56	.57
	15	1/2 inch.	1.35	78	22	10.54	.60		223	do.	1.46	86	14	8.90	.64
	16	do.	1.45	87	13	11.18	.56		224	do.	1.55	91	9	9.61	.64
19A	33	do.	1.35	51	49	6.63	.77	20	173	do.	1.35	45	55	7.18	.46
	34	do.	1.40	59	41	8.19	.77		174	do.	1.41	60	40	8.81	.51
	35	do.	1.45	70	30	9.66	.81		175	do.	1.45	66	34	9.54	.48
	36	do.	1.54	76	24	12.82	.72		176	do.	1.54	76	24	10.47	.51
19C	41	1/2 inch.	1.35	46	54	6.66	.72	27	229	do.	1.35	65	35	8.22	.65
	42	do.	1.40	56	44	10.02	.68		230	do.	1.40	87	13	9.33	.65
	43	do.	1.45	68	32	10.20	.71		231	do.	1.46	90	10	10.12	.63
	44	do.	1.54	74	26	13.21	.71		232	do.	1.55	95	5	10.47	.56
22	37	1/2 inch.	1.35	67	33	6.13	.77	30	233	do.	1.35	77	23	6.79	.46
	38	do.	1.40	75	25	7.23	.78		234	do.	1.40	77	23	7.66	.46
	39	do.	1.45	81	19	8.07	.71		235	do.	1.46	90	10	8.54	.44
	40	do.	1.55	82	18	9.12	.78		236	do.	1.55	92	8	9.32	.47
21B	117	do.	1.34	16	84	5.95	2.34	29	253	do.	1.35	86	14	6.38	.48
	118	do.	1.40	23	77	7.83	2.41		254	do.	1.40	92	8	5.88	.52
	119	do.	1.46	27	73	9.52	2.48		255	do.	1.46	92	8	6.54	.52
	120	do.	1.52	33	67	10.06	2.59		256	do.	1.55	92	8	7.70	.48
22	121	1/2 inch.	1.35	24	76	6.99	2.34	30	257	do.	1.35	81	19	7.36	.45
	122	do.	1.40	33	67	7.93	2.52		258	do.	1.40	85	15	6.47	.47
	123	do.	1.46	40	60	9.10	2.52		259	do.	1.46	85	15	7.69	.45
	124	do.	1.51	45	55	9.30	2.62		260	do.	1.55	92	8	8.34	.44
21B	157	do.	1.34	83	17	7.98	.47	24	269	1/2 inch.	1.35	29	71	5.91	.53
	158	do.	1.40	90	10	8.98	.46		270	do.	1.40	42	58	6.89	.56
	159	do.	1.45	91	9	8.47	.49		271	do.	1.46	50	50	7.98	.57
	160	do.	1.54	91	9	9.25	.40		272	do.	1.55	53	47	11.70	.54
22	161	1/2 inch.	1.34	85	15	7.43	.45	32	277	1/2 inch.	1.35	76	24	4.42	1.47
	162	do.	1.40	90	10	8.11	.56		278	do.	1.40	84	16	5.75	1.54
	163	do.	1.45	93	7	8.38	.48		279	do.	1.46	93	7	7.53	1.53
	164	do.	1.54	95	5	9.18	.57		280	do.	1.55	93	7	7.05	1.63
22	149	1/2 inch.	1.35	73	27	5.01	2.75	31	281	do.	1.35	64	36	5.83	.75
	150	do.	1.40	80	20	6.80	3.00		282	do.	1.40	73	27	6.35	.81
	151	do.	1.45	80	20	6.81	3.15		283	do.	1.46	80	20	8.20	.75
	152	do.	1.54	88	12	7.31	3.47		284	do.	1.55	84	14	8.62	.81
22	153	1/2 inch.	1.35	73	27	3.78	2.34	33	293	do.	1.35	93	7	2.20	.89
	154	do.	1.41	82	18	4.43	2.65		294	do.	1.40	95	5	2.55	1.00
	155	do.	1.45	83	17	5.17	2.76		295	do.	1.46	93	7	2.98	.98
	156	do.	1.54	87	13	5.30	3.03		296	do.	1.55	98	2	3.69	.97

a As received.

Refuse float-and-sink tests.

Denver No.	Refuse from wash- ing test No.	Test No.	Specific gravity of solution used.	Percentage of—		Float-coal analyses (dry basis).		
				Float.	Sink.	Ash.	Sulphur.	
15A	232	17	1.37	69	31	7.00	1.08	
		18	1.41	79	21	8.20	1.18	
	233	19	1.47	88	12	11.09	1.08	
		20	1.58	90	10	12.03	1.24	
	234	21	1.35	66	34	6.94	1.08	
		22	1.40	71	29	8.85	1.23	
	235	23	1.45	78	22	9.91	1.05	
		24	1.54	85	15	13.02	1.08	
	15B	236	25	1.35	60	40	6.07	1.06
			26	1.40	65	35	7.88	1.98
237		27	1.45	67	33	8.65	1.04	
		28	1.54	75	25	12.25	1.04	
238		29	1.35	37	63	6.75	.95	
		30	1.40	44	56	9.57	1.02	
239		31	1.45	59	41	10.33	.07	
		32	1.54	57	43	13.75	1.08	
17		240	33	1.32	33	67	5.00	.92
			34	1.40	33	67	5.09	.99
	241	35	1.45	41	59	7.42	1.11	
		36	1.54	50	50	9.92	1.42	
	242	37	1.35	26	74	7.41	.84	
		38	1.40	41	59	10.21	.78	
	243	39	1.46	57	43	12.26	.70	
		40	1.55	66	34	13.58	.72	
	18	244	41	1.32	3	97	Destroyed.	
			42	1.40	5	95	8.39	.90
245		43	1.45	7	93	10.34	1.07	
		44	1.54	7	93	13.45	1.06	
246		45	1.32	20	80	4.89	.91	
		46	1.40	30	70	6.28	.96	
247		47	1.45	36	64	7.36	1.04	
		48	1.54	36	64	8.57	1.03	
19A		248	49	1.32	12	88	4.96	1.02
			50	1.40	18	82	7.57	1.03
	249	51	1.45	22	78	7.32	1.06	
		52	1.54	23	77	9.67	1.06	
	250	53	1.35	20	80	6.99	.65	
		54	1.40	38	62	11.13	.73	
	251	55	1.46	43	57	14.44	.77	
		56	1.55	50	50	17.50	.53	
	19B	252	57	1.35	16	84	6.65	.64
			58	1.40	30	70	11.77	.67
253		59	1.46	39	61	14.46	.69	
		60	1.55	56	44	18.15	.62	
254		61	1.35	24	76	7.97	.61	
		62	1.40	31	69	12.84	.57	
255		63	1.46	47	53	12.44	.69	
		64	1.55	50	50	15.84	.52	
20		256	65	1.35	21	79	9.55	.90
			66	1.40	38	62	12.92	.70
	257	67	1.46	51	49	14.18	.63	
		68	1.55	65	35	17.99	.60	
	258	69	1.35	13	87	9.78	.55	
		70	1.40	20	80	12.04	.42	
	259	71	1.46	24	76	14.74	.63	
		72	1.55	33	67	18.78	.45	
	21A	260	73	1.35	41	59	7.96	.68
			74	1.40	48	52	10.30	.58
261		75	1.46	47	53	12.44	.69	
		76	1.55	50	50	15.84	.52	
262		77	1.35	21	79	9.55	.90	
		78	1.40	38	62	12.92	.70	
263		79	1.46	51	49	14.18	.63	
		80	1.55	65	35	17.99	.60	
21B		264	81	1.35	13	87	9.78	.55
			82	1.40	20	80	12.04	.42
	265	83	1.46	24	76	14.74	.63	
		84	1.55	33	67	18.78	.45	
	266	85	1.35	41	59	7.96	.68	
		86	1.40	48	52	10.30	.58	
	267	87	1.46	56	44	11.79	.54	
		88	1.55	63	37	13.02	.51	
	22	268	89	1.35	30	70	6.26	.65
			90	1.40	50	50	8.64	.67
269		91	1.46	60	40	11.24	.62	
		92	1.55	69	31	11.95	.54	
270		93	1.35	19	81	7.91	.55	
		94	1.40	26	74	11.76	.54	
271		95	1.46	42	58	13.67	.51	
		96	1.55	53	47	16.18	.51	
23		272	97	1.35	29	71	6.02	.56
			98	1.40	47	53	8.27	.72
	273	99	1.46	61	39	10.98	.64	
		100	1.54	70	30	14.45	.95	
	274	101	1.35	18	82	7.25	.75	
		102	1.40	29	71	9.98	.77	
	275	103	1.46	41	59	13.03	.74	
		104	1.54	51	49	15.83	.69	
	24	276	105	1.35	14	86	6.59	.94
			106	1.40	25	75	8.20	1.18
277		107	1.46	31	69	11.09	1.08	
		108	1.54	43	57	12.03	1.24	
278		109	1.35	20	80	6.94	1.08	
		110	1.40	31	69	8.85	1.23	
279		111	1.45	44	56	9.91	1.05	
		112	1.54	53	47	13.02	1.08	
25		280	113	1.35	38	62	6.07	1.06
			114	1.40	6	54	7.88	1.98
	281	115	1.46	54	46	8.65	1.04	
		116	1.54	58	42	12.25	1.04	
	282	117	1.35	6	94	6.75	.95	
		118	1.40	11	89	9.57	1.02	
	283	119	1.45	12	88	10.33	.07	
		120	1.51	14	86	13.75	1.08	
	26	284	121	1.34	6	94	5.00	.92
			122	1.41	11	89	9.09	.99
285		123	1.45	12	88	9.92	1.11	
		124	1.51	14	86	12.26	1.42	
286		125	1.34	6	94	6.75	.95	
		126	1.41	11	89	9.57	1.02	
287		127	1.45	12	88	10.33	.07	
		128	1.51	14	86	13.75	1.08	
27		288	129	1.34	6	94	5.00	.92
			130	1.41	11	89	9.09	.99
	289	131	1.45	14	86	9.92	1.11	
		132	1.51	15	85	12.26	1.42	
	290	133	1.35	11	89	6.75	.95	
		134	1.40	18	82	9.57	1.02	
	291	135	1.45	22	78	10.33	.07	
		136	1.54	27	73	13.75	1.08	
	28	292	137	1.35	35	65	6.75	.95
			138	1.40	48	52	8.65	1.04
293		139	1.45	46	54	9.92	1.11	
		140	1.55	55	45	12.26	1.42	
294		141	1.35	53	47	6.75	.95	
		142	1.40	59	41	9.57	1.02	
295		143	1.45	61	39	10.33	.07	
		144	1.55	68	32	13.75	1.08	
29		296	145	1.35	40	60	6.75	.95
			146	1.40	46	54	8.65	1.04
	297	147	1.45	57	43	9.92	1.11	
		148	1.55	58	42	12.26	1.42	
	298	149	1.35	36	64	6.75	.95	
		150	1.40	48	52	8.65	1.04	
	299	151	1.45	58	42	9.92	1.11	
		152	1.54	67	33	13.75	1.08	
	30	300	153	1.35	36	64	6.75	.95
			154	1.40	48	52	8.65	1.04
301		155	1.45	58	42	9.92	1.11	
		156	1.54	67	33	13.75	1.08	
302		157	1.35	36	64	6.75	.95	
		158	1.40	48	52	8.65	1.04	
303		159	1.45	58	42	9.92	1.11	
		160	1.54	67	33	13.75	1.08	
31		304	161	1.35	36	64	6.75	.95
			162	1.40	48	52	8.65	1.04
	305	163	1.45	58	42	9.92	1.11	
		164	1.54	67	33	13.75	1.08	
	306	165	1.35	36	64	6.75	.95	
		166	1.40	48	52	8.65	1.04	
	307	167	1.45	58	42	9.92	1.11	
		168	1.54	67	33	13.75	1.08	
	32	308	169	1.35	36	64	6.75	.95
			170	1.40	48	52	8.65	1.04
309		171	1.45	58	42	9.92	1.11	
		172	1.54	67	33	13.75	1.08	
310		173	1.35	36	64	6.75	.95	
		174	1.40	48	52	8.65	1.04	
311		175	1.45	58	42	9.92	1.11	
		176	1.54	67	33	13.75	1.08	
33		312	177	1.35	36	64	6.75	.95
			178	1.40	48	52	8.65	1.04
	313	179	1.45	58	42	9.92	1.11	
		180	1.54	67	33	13.75	1.08	
	314	181	1.35	36	64	6.75	.95	
		182	1.40	48	52	8.65	1.04	
	315	183	1.45	58	42	9.92	1.11	
		184	1.54	67	33	13.75	1.08	
	34	316	185	1.35	36	64	6.75	.95
			186	1.40	48	52	8.65	1.04
317		187	1.45	58	42	9.92	1.11	
		188	1.54	67	33	13.75	1.08	
318		189	1.35	36	64	6.75	.95	
		190	1.40	48	52	8.65	1.04	
319		191	1.45	58	42	9.92	1.11	
		192	1.54	67	33	13.75	1.08	
35		320	193	1.35	36	64	6.75	.95
			194	1.40	48	52	8.65	1.04
	321	195	1.45	58	42	9.92	1.11	
		196	1.54	67	33	13.75	1.08	
	322	197	1.35	36	64	6.75	.95	
		198	1.40	48	52	8.65	1.04	
	323	199	1.45	58	42	9.92	1.11	
		200	1.54	67	33	13.75	1.08	
	36	324	201	1.35	36	64	6.75	.95
			202	1.40	48	52	8.65	1.04
325		203	1.45	58	42	9.92	1.11	
		204	1.54	67	33	13.75	1.08	
326		205	1.35	36	64	6.75	.95	
		206	1.40	48	52	8.65	1.04	
327		207	1.45	58	42	9.92	1.11	
		208	1.54	67	33	13.75	1.08	
37		328	209	1.35	36	64	6.75	.95
			210	1.40	48	52	8.65	1.04
	329	211	1.45	58	42	9.92	1.11	
		212	1.54	67	33	13.75	1.08	
	330	213	1.35	36	64	6.75	.95	
		214	1.40	48	52	8.65	1.04	
	331	215	1.45	58	42	9.92	1.11	
		216	1.54	67	33	13.75	1.08	
	38	332	217	1.35	36	64	6.75	.95
			218	1.40	48	52	8.65	1.04
333		219	1.45	58	42	9.92	1.11	
		220	1.54	67	33	13.75	1.08	
334		221	1.35	36	64	6.75	.95	
		222	1.40	48	52	8.65	1.04	
335		223	1.45	58	42	9.92	1.11	
		224	1.54	67	33	13.75	1.08	
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Refuse float-and-sink tests—Continued.

Denver No.	Refuse from washing test No.	Test No.	Specific gravity of solution used.	Percentage of—		Float-coal analyses (dry basis).		Denver No.	Refuse from washing test No.	Test No.	Specific gravity of solution used.	Percentage of—		Float-coal analyses (dry basis).	
				Float.	Sink.	Ash.	S u l - phur.					Float.	Sink.	Ash.	S u l - phur.
21B	270	225	1.35	53	47	8.11	0.57	24	276	261	1.35	13	87	5.96	0.59
		226	1.40	60	40	9.27	.51			262	1.40	21	79	7.90	.56
		227	1.46	65	35	9.74	.50			263	1.46	23	77	10.86	.57
		228	1.55	70	30	11.36	.53			264	1.55	31	69	13.87	.56
27	271	237	1.35	32	68	7.50	.66	27	277	273	1.35	7	93	7.51	.59
		238	1.40	44	56	10.22	.58			274	1.40	10	90	10.04	.63
		239	1.46	56	44	12.82	.64			275	1.46	16	84	12.24	.57
		240	1.55	63	37	16.28	.61			276	1.55	17	83	17.72	.52
28	272	241	1.35	20	80	6.65	.66	32	278	285	1.35	18	82	4.73	1.41
		242	1.40	27	73	10.40	.59			286	1.40	30	70	7.15	1.28
		243	1.46	43	57	12.41	.58			287	1.46	33	67	7.89	1.28
		244	1.55	61	39	17.57	.56			288	1.55	45	55	11.78	1.69
30	273	245	1.35	26	74	8.26	.46	31	279	289	1.35	13	87	4.99	.81
		246	1.40	34	66	9.43	.46			290	1.40	16	84	8.25	.80
		247	1.46	42	58	11.93	.48			291	1.46	16	84	9.83	1.00
		248	1.55	51	49	15.26	.42			292	1.55	25	75	12.05	.94
29	274	249	1.35	52	48	5.34	.48	33	280	297	1.35	30	70	4.02	.93
		250	1.40	62	38	7.13	.51			298	1.40	38	62	5.26	.91
		251	1.46	67	33	7.97	.47			299	1.46	43	57	5.59	1.04
		252	1.55	78	22	9.87	.46			300	1.55	43	57	8.88	.94
29	275	265	1.35	34	66	6.62	.50	30	275	266	1.40	42	58	8.70	.42
		266	1.46	46	54	8.85	.46			267	1.46	46	54	8.85	.46
		267	1.46	46	54	8.85	.46			268	1.55	60	40	14.96	.61
		268	1.55	60	40	14.96	.61								

Raw-coal screening tests.^a

Denver No.	Percentage of coal passing screen.							Denver No.	Percentage of coal passing screen.						
	Over 1½-inch.	Through 1½-inch and over 1-inch.	Through 1-inch and over ¾-inch.	Through ¾-inch and over ½-inch.	Through ½-inch and over ¼-inch.	Through ¼-inch and over ⅛-inch.	Through ⅛-inch.		Over 1½-inch.	Through 1½-inch and over 1-inch.	Through 1-inch and over ¾-inch.	Through ¾-inch and over ½-inch.	Through ½-inch and over ¼-inch.	Through ¼-inch and over ⅛-inch.	Through ⅛-inch.
15A	3	7	22	23	12	11	22	23	16	18	3	10	87
15B	7	7	8	16	9	11	42		9	14	25
17	3	19	19	6	12	41	32	35	24	19	8	6	8
16	3	10	22	12	19	34	33	17	20	28	8	10	17
17	1	1	12	86	20	2	14	7	21	56
	3	5	5	17	70	25	3	15	10	17	55
19A	2	12	21	9	16	40	27	3	15	32	11	15	24
19C	3	97	28	6	14	20	9	17	34
	5	11	26	11	13	34	29	2	13	22	7	13	43
22	3	3	6	88	30	2	9	21	7	14	47
23	11	15	19	7	11	37								

^a These tests show the result of screening after the samples had been passed through a 24 by 30 inch corrugated-roll disintegrator (three-fourths inch between the rolls), or, for the three tests indicated, through a Pennsylvania hammer crusher.

^b Hammer crusher.

NOTES ON TESTS.

Denver No. 15A is an Illinois run-of-mine coal of good quality. Four tests were made, the percentage of refuse varying from 8 to 25 per cent, the loss of coal increasing greatly as the percentage of refuse was increased, while the washed-coal analysis remained about the same for all tests, indicating no gain in a separation giving over 8 per cent refuse. The washing tests compare favorably with the raw-coal float-and-sink tests.

Denver No. 15B is a slack coal from the same Illinois mine and, as will be noted, carried a higher percentage of ash. One test was made on the special jig and three on the Richards. The special-jig test gave better results, but the loss of coal was higher than in those made on the Richards jig. The refuse from the Richards jig was very free from coal, but on account of the then existing shape of the refuse gates it was not possible to attain so high a percentage of refuse as was desired from this machine in these tests. The shape of these gates was changed after the tests on this coal were made, and thereafter they permitted a higher percentage of refuse to be discharged. Test 236 on the special jig compared favorably with the raw-coal float-and-sink tests. The others were not so good.

Denver No. 16 is from the Sopris mine (Sopris, Colo.), and may be compared with that from the Francisco mine (Sopris, Colo.), which was tested the previous year under the designation Denver No. 7. Comparison of the tests on these two coals shows that the analyses of the resulting washed coals were about the same and the coal loss was high in both. The tests on the Richards jig were not quite so good as the one made on the special jig, and this test compares very well with the raw-coal float-and-sink tests.

For Denver No. 17 the test on the special jig was the most satisfactory and compares well with the float-and-sink tests. The Richards jig tests gave a cleaner refuse, but not enough of it.

For Denver No. 18 the tests on the special jig were good and compare favorably with the float-and-sink tests, though the coal loss is a little too high. Test 245 on the Richards jig was very good and the coal loss low.

The ash reduction of the tests on Denver No. 19A do not compare very well with the float-and-sink tests and the coal losses are all very high. Forty per cent of the coal as washed, however, passed through a $\frac{1}{4}$ -inch mesh screen, and more careful washing will probably show much better results.

In Denver No. 19C there was only sufficient coal for one test, and this one did not give satisfactory results when compared with the float-and-sink tests.

The test on Denver No. 21A was quite satisfactory in all respects.

On Denver No. 21B special-jig tests gave best results, though the float-and-sink tests would indicate that a little better separation is possible.

Denver No. 22 is a very high ash coal from Montana, the raw coal analyzing 38.97 per cent ash and 7.93 per cent sulphur, as against an analysis of 32.19 per cent ash and 0.54 per cent sulphur for the Montana coal tested last year (Denver No. 5). However, the tests made on the special jig on Denver No. 22 were much better than those of last year on Denver No. 5; in fact, test 255 is very

satisfactory. The loss of coal is high, but in view of the fact that over 50 per cent of the run-of-mine coal must be discarded as refuse to reduce the ash as desired this coal loss is not bad, for the coal was very fine and very hard to handle in the washer plant. It will be noted, by referring to the table of screening tests, that 88 per cent of this run-of-mine coal passed through a $\frac{1}{4}$ -inch mesh screen.

The tests on Denver No. 23, except the one made on the Richards jig, are very good, although the coal losses were very high; otherwise they compare favorably with the float-and-sink tests. With a much longer sludge-recovery tank the loss of coal could be brought down to a more reasonable figure.

The tests on Denver No. 20 compare fairly well with the float-and-sink tests, though they could be slightly improved with more careful jig adjustment.

The tests on Denver No. 25 are fair, but, like those on Denver No. 20, could be improved.

Of all the other coals tested and not yet mentioned, but one test was made on each; nothing need be added to the results in the tables.

Denver No. 24 is a refuse from one of the washeries of the Trinidad district. A few tests were made on it with the view of determining the possibility of the use of the Richards jig as it now stands for rewashing; but the tests were not satisfactory, as will be noted from the results appearing in the tables.

COKING TESTS.

By A. W. BELDEN.

EQUIPMENT.

The equipment of the coking section proved so satisfactory in the previous tests that no change was deemed necessary. The equipment included a battery of two beehive ovens (one 7 feet high by 12 feet in diameter, the other 6 feet 3 inches high by 12 feet in diameter), a standard larry of 8 tons capacity, and the necessary scales for accurate weighing of coal charged and coke produced.

PROCEDURE OF TESTS.

All coal was finely crushed through a Pennsylvania hammer crusher, except as otherwise noted in the detailed report of the tests. The proportionate sizes of the coal after being put through the hammer crusher varied somewhat, depending on the friability of the coal, but the average, taken from a large number of samples, showed

practically the same as in former work, as follows: Through $\frac{1}{8}$ -inch mesh, 100 per cent; over 10-mesh, 31.43 per cent; over 20-mesh, 24.29 per cent; over 40-mesh, 22.86 per cent; over 60-mesh, 11.42 per cent; through 60-mesh, 10 per cent. The sampling of coal and coke and the handling of the ovens were practically the same as described in Bulletin 336 of the United States Geological Survey, from which the following paragraphs are quoted:

Both the door and the trunnel head of the oven were always closed directly after the oven was drawn and it was allowed to gather heat, the length of time varying as necessity demanded. The average time was one and one-half hours.

The sample of coal was taken at regular intervals, as the charge was emptied from bin to larry, by means of a small shovel holding about one-fourth pound. The total weight of the sample averaged 45 pounds.

The sample of coke was taken from five different parts of the oven, as nearly as possible from the same location for each test, as follows: 2 feet from the oven door; 2 feet from each side, on a line drawn from the center of the oven; at the center; and 2 feet from the back wall, on a line with the point of selection of the pieces taken from the door and the center. The separate pieces of coke extended the whole height of the charge and were as nearly uniform in size as possible.

NUMBER OF TESTS.

From August 21, 1908, to January 2, 1909, 69 tests were made on 22 coals from five States and one Territory, as shown in the following tables. Of these tests, 12 were made on raw coal, 55 on washed coal, and 2 on mixtures of washed coals. Of the 22 different coals, five (Nos. 19, 26, 28, 32, and 33) produced no coke, whether tested raw or washed, crushed or not crushed.

TABULATED RESULTS.

The results of the coking tests will be found in the detailed reports on the several samples presented herewith. The method hitherto used to state all the items, so as to show the yield of dry coke from dry coal, the coke as received from coal as charged, etc., has been continued. The percentage of coke remaining on a screen with 2-inch mesh, after four consecutive 6-foot drops without intermediate screening, as well as the percentage after each drop, is given in the last item under "Physical properties of coke." The first four items represent the percentage from each separate drop with all material less than 2-inch screened out, the fifth item the percentage after four consecutive drops, all material being returned each time. It was decided to make the phosphorus determination on each coke produced in order that a better average might be obtained. "Cell structure" refers to the general appearance as to size and not to the number of cells as given by percentage of cells by volume. In many tests in which the cell structure as determined from general

appearance is small the percentage of cells by volume is quite the reverse. The following abbreviations are used in the tables:

scr., screenings.	r. o. m., run of mine.
f. c., finely crushed.	sl., slack.
n. c., not crushed.	w., washed.
r., raw.	

DENVER No. 15A.

Coking tests.

	Test—				
	247.	248.	249.	250.	251.
Date.....	8, 24, 08	8, 28, 08	8, 31, 08	9, 1, 08	9, 2, 08
Duration..... hours..	48	52	57	60	57
Size:					
As shipped.....	r. o. m.	r. o. m.	r. o. m.	r. o. m.	r. o. m.
As used.....	r., f. c.	w., f. c.	w., f. c.	w., f. c.	w., f. c.
Coal charged:					
Wet..... pounds..	11,730	11,000	11,800	12,200	12,850
Dry..... do.....	10,833	9,903	10,651	11,551	11,664
Coke produced:					
Wet..... do.....	7,200	6,229	6,302	6,664	7,496
..... per cent..	61.38	56.63	53.40	54.62	58.33
Dry..... pounds..	7,019	6,165	6,264	6,620	7,458
..... per cent..	64.79	62.25	58.81	57.31	63.94
Breeze produced:					
Wet..... pounds..	400	438	389	458	512
..... per cent..	3.41	3.98	3.30	3.75	3.99
Dry..... pounds..	390	434	387	455	509
..... per cent..	3.60	4.38	3.63	3.94	4.38
Total yield:					
Wet..... do.....	64.79	60.61	56.70	58.37	62.32
Dry..... do.....	68.39	66.63	62.44	61.25	68.30
Physical properties of coke:					
Specific gravity—					
Apparent.....	0.83	0.76	0.77	0.78	0.82
Real.....	1.82	1.78	1.77	1.79	1.79
Volume—					
Coke..... per cent..	46.00	43.00	44.00	44.00	46.00
Cells..... do.....	54.00	57.00	56.00	56.00	54.00
Weight per cubic foot—					
Wet..... pounds..	84.16	82.45	82.64	83.29	84.54
Dry..... do.....	50.48	46.90	47.70	48.35	50.86
6-foot drop test over 2-inch mesh—					
1..... per cent..	98.00	98.00	98.50	98.50	98.00
2..... do.....	93.00	95.50	98.00	98.00	95.50
3..... do.....	89.50	92.00	94.00	95.50	92.00
4..... do.....	85.50	89.50	91.50	91.00	89.00
5..... do.....	89.00	92.00	90.00	92.50	91.50

Remarks.—Test 247: Light gray and silvery. Breakage good; long, large pieces. Cell structure good. Metallic ring. Good, strong coke.

Test 248: Light gray, some little silvery deposit of carbon. Somewhat fingered. One-inch layer of black, soft coke on bottom of oven, due to hot bottom. Cell structure good. Metallic ring.

Test 249: Light gray and silvery. Cell structure good. Breakage somewhat fingered, but pieces of good uniform size. Metallic ring.

Test 250: Light gray, some little silvery deposit of carbon. Breakage same as test 249. Metallic ring. Good coke.

Test 251: Light gray and silvery. Cell structure good. Breakage same as test 249. Metallic ring. Good coke. Washing of this coal results in change of coke from good, long, large pieces to somewhat fingered condition.

Chemical analyses.

Test No.	Laboratory No.		Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Phosphorus.
247	482-D...	Coal	Wet.. 7.65	33.97	49.60	8.78	0.91
			Dry..	36.78	53.71	9.51	.99
	487-D...	Coke	Wet.. 2.52	2.20	79.63	15.65	.86	0.0203
			Dry..	2.26	81.68	16.06	.88
248	492-D...	Coal	Wet.. 9.97	34.82	48.70	6.51	1.01
			Dry..	38.68	54.09	7.23	1.12
	498-D...	Coke	Wet.. 1.02	1.58	87.38	10.02	.80	.0165
			Dry..	1.60	88.28	10.12	.82
249	496-D...	Coal	Wet.. 9.74	34.70	49.35	6.21	.97
			Dry..	38.45	54.67	6.88	1.08
	500-D...	Coke	Wet.. .61	1.79	86.77	10.83	.81	.0173
			Dry..	1.80	87.31	10.89	.82
250	497-D...	Coal	Wet.. 5.32	38.43	49.42	6.83	1.02
			Dry..	40.60	52.19	7.21	1.08
	504-D...	Coke	Wet.. .66	1.76	86.44	11.14	.71	.0152
			Dry..	1.77	87.01	11.22	.72
251	499-D...	Coal	Wet.. 9.23	35.64	48.30	6.83	1.00
			Dry..	39.25	53.23	7.52	1.10
	507-D...	Coke	Wet.. .51	1.80	87.17	10.52	.78	.0166
			Dry..	1.81	87.61	10.58	.78

DENVER No. 15B.*Coking tests.*

	Test—			
	252.	254.	256.	258.
Date.....	9, 3, 08	9, 6, 08	9, 12, 08	9, 14, 08
Duration..... hours..	55	52	66	51
Size:				
As shipped.....	scr.	scr.	scr.	scr.
As used.....	w., f. c.	w., f. c.	w., f. c.	w., f. c.
Coal charged:				
Wet..... pounds..	11,980	11,190	12,220	11,630
Dry..... do.....	10,638	8,739	10,300	10,724
Coke produced:				
Wet..... do.....	6,068	5,234	5,940	6,600
..... per cent..	56.65	46.77	48.61	56.75
Dry..... pounds..	6,035	5,153	5,806	6,584
..... per cent..	56.73	58.97	56.36	61.40
Breeze produced:				
Wet..... pounds..	359	414	510	350
..... per cent..	3.00	3.70	4.17	3.01
Dry..... pounds..	357	408	498	349
..... per cent..	3.36	4.67	4.84	3.25
Total yield:				
Wet..... do.....	53.65	50.47	52.78	59.76
Dry..... do.....	60.09	63.64	61.20	64.65
Physical properties of coke:				
Specific gravity—				
Apparent.....	0.81	0.77	0.81	0.86
Real.....	1.78	1.80	1.79	1.80
Volume—				
Coke..... per cent..	46.00	43.00	45.00	48.00
Cells..... do.....	54.00	57.00	55.00	52.00
Weight per cubic foot—				
Wet..... pounds..	83.93	82.83	83.67	85.95
Dry..... do.....	50.25	47.28	49.38	53.53
6-foot drop test over 2-inch mesh—				
1..... per cent..	98.00	98.00	98.50	98.50
2..... do.....	96.50	94.50	95.00	98.00
3..... do.....	95.00	93.00	92.50	94.00
4..... do.....	90.00	90.00	91.00	91.50
5..... do.....	93.00	91.50	91.50	95.50

Remarks.—Test 252: Light gray and silvery. Cell structure good. Breakage somewhat fingered, but pieces of good uniform size. Metallic ring. Good coke.

Test 254: Light gray with large deposit of carbon. Breakage, good; long, large, uniform-sized pieces, same as from test 247, Denver 15A, raw coal. Probably due to ash in coal not reduced so low as in other washed coal. Metallic ring. Good coke.

Test 256: Light gray and silvery. Cell structure good. Breakage somewhat fingered, but pieces of good, uniform size. Metallic ring. One-half inch black butts accounts for high volatile in coke.

Test 258: Light gray and silvery. Cell structure good. Somewhat fingered, but pieces of good uniform size. Metallic ring. Good coke.

Chemical analyses.

Test No.	Laboratory No.			Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Phosphorus.
252	503-D.	Coal	Wet	11.19	35.11	47.27	6.43	0.97
			Dry		39.53	53.23	7.24	1.09
	508-D.	Coke	Wet	.55	1.40	87.44	10.61	.80	0.0170
254	506-D.	Coal	Wet	21.01	1.41	87.91	10.66	.80
			Dry		29.81	41.54	8.14	.90
	517-D.	Coke	Wet	1.54	37.10	82.59	10.31	1.14
256	516-D.	Coal	Wet	14.89	2.14	80.55	15.77	.93	.0206
			Dry		2.17	81.81	16.02	.95
	524-D.	Coke	Wet	2.27	31.95	45.73	7.43	.87
258	534-D.	Coal	Wet	7.79	37.55	53.72	8.73	1.02
			Dry		4.81	78.73	14.19	1.04	.0198
	535-D.	Coke	Wet	.24	4.92	80.56	14.52	1.06
258	542-D.	Coal	Wet		32.56	50.35	9.30	.91
		Coke	Dry		35.31	54.60	10.09	.99
			Wet		1.20	82.70	15.86	.81	.0146
			Dry		1.20	82.90	15.90	.81

DENVER No. 16.

Coking tests.

	Test—				
	259.	260.	266.	268.	271.
Date.....	9, 17, 08	9, 18, 08	9, 26, 08	9, 30, 08	10, 5, 08
Duration.....hours	45	49	56	50	72
Size:					
As shipped.....	r. o. m.	r. o. m.	r. o. m.	r. o. m.	r. o. m.
As used.....	w., f. c.	w., f. c.	w., f. c.	w., f. c.	w., f. c.
Coal charged:					
Wet.....pounds	11,630	13,630	14,760	11,570	15,150
Dry.....do	11,139	12,782	13,139	10,688	13,668
Coke produced:					
Wet.....do	8,100	8,580	9,300	7,500	9,669
.....per cent.	69.65	62.73	63.01	65.19	63.82
Dry.....pounds	8,081	8,467	9,282	7,488	9,621
.....per cent.	72.55	66.24	70.65	70.06	70.39
Breeze produced:					
Wet.....pounds	269	300	284	237	340
.....per cent.	2.31	2.20	1.92	1.99	2.24
Dry.....pounds	268	297	283	237	338
.....per cent.	2.41	2.32	2.15	2.22	2.47
Total yield:					
Wet.....do	71.96	64.93	64.93	65.18	66.06
Dry.....do	74.96	68.56	72.80	72.28	72.86
Physical properties of coke:					
Specific gravity—					
Apparent.....	1.03	1.09	1.09	1.07	1.08
Real.....	1.88	1.92	1.89	1.93	1.90
Volume—					
Coke.....per cent.	55.00	57.00	52.00	55.00	57.00
Cells.....do	45.00	43.00	48.00	45.00	43.00
Weight per cubic foot—					
Wet.....pounds	92.16	94.15	97.80	94.72	93.84
Dry.....do	64.08	67.32	67.86	66.64	67.02
6-foot drop test over 2-inch mesh—					
1.....per cent.	96.00	96.00	99.60	98.00	96.50
2.....do	94.50	95.50	97.00	96.50	96.50
3.....do	93.00	93.00	95.50	94.00	95.00
4.....do	91.50	90.80	93.50	91.50	91.50
5.....do	93.50	93.50	94.00	92.00	93.50

Remarks.—Test 259: Light gray and silvery. Breakage good, large pieces. Cell structure small. Good, hard, strong coke.

Test 260: Light gray and silvery. Breakage, good-sized pieces, somewhat cross fractured, but uniform. Cell structure small. Good, strong, hard coke.

Test 266: Light gray and silvery. Breakage somewhat irregular, but pieces of good, large, uniform size. Cell structure small, dense. Good, hard, strong, heavy coke.

Test 268: Light gray and silvery. Breakage irregular, but pieces of good, large, uniform size. Cell structure small, dense. Good, strong, hard coke.

Test 271: Light gray and silvery, large deposit of carbon. Breakage good, long, large pieces. Cell structure small, dense. Metallic ring. Good, strong, hard, heavy coke.

Chemical analyses.

Test No.	Laboratory No.		Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Phosphorus.
259	540-D	Coal	Wet.. 4.22	29.95	54.56	11.27	0.62
			Dry.....	31.27	56.95	11.78	.65
	544-D	Coke	Wet.. .24	1.11	81.78	16.87	.58	0.0173
			Dry.....	1.11	81.98	16.91	.58
260	546-D	Coal	Wet.. 6.22	30.07	53.03	10.68	.66
			Dry.....	32.07	56.54	11.39	.70
	548-D	Coke	Wet.. .97	.92	81.75	16.36	.64	.0190
			Dry.....	.93	82.55	16.52	.65
266	555-D	Coal	Wet.. 10.98	28.33	49.65	11.04	.59
			Dry.....	31.82	55.78	12.40	.66
	565-D	Coke	Wet.. .19	.82	82.18	16.81	.52	.0296
			Dry.....	.82	82.34	16.84	.52
268	562-D	Coal	Wet.. 9.96	28.41	50.84	10.79	.62
			Dry.....	31.55	56.47	11.98	.69
	590-D	Coke	Wet.. .16	.96	81.82	17.06	.58	.0291
			Dry.....	.96	81.95	17.09	.59
271	589-D	Coal	Wet.. 9.78	27.84	51.27	11.11	.66
			Dry.....	30.85	56.84	12.31	.73
	616-D	Coke	Wet.. .50	1.12	81.87	16.51	.42	.0352
			Dry.....	1.13	82.27	16.60	.42

DENVER No. 17.

Coking tests.

	Test—			
	253.	255.	257.	261.
Date.....	9,5,08	9,8,08	9,13,08	9,20,08
Duration..... hours	47	53	42	60
State:				
As shipped.....	r. o. m.	r. o. m.	r. o. m.	r. o. m.
As used.....	w., f. c.	w., f. c.	w., f. c.	w., f. c.
Coal charged:				
Wet..... pounds	11,920	12,220	11,200	13,700
Dry..... do	11,279	11,196	10,489	12,805
Coke produced:				
Wet..... { do	7,650	7,650	6,840	8,850
per cent.....	64.18	62.90	61.07	64.60
Dry..... { pounds	7,617	7,522	6,806	8,800
per cent.....	67.53	67.18	64.89	68.72
Breeze produced:				
Wet..... { pounds	540	375	375	600
per cent.....	4.53	3.07	3.35	4.38
Dry..... { pounds	538	374	373	597
per cent.....	4.77	3.34	3.56	4.06
Total yield:				
Wet..... per cent	68.71	65.67	64.42	68.98
Dry..... do	72.30	70.52	68.45	73.38
Physical properties of coke:				
Specific gravity—				
Apparent.....	1.06	0.99	1.00	1.09
Real.....	1.93	1.89	1.91	1.94
Volume—				
Coke..... per cent	55.00	52.00	52.00	56.00
Cells..... do	45.00	48.00	48.00	44.00
Weight per cubic foot—				
Wet..... pounds	93.92	91.48	91.97	95.02
Dry..... do	65.84	61.53	62.03	67.59
6-foot drop test over 2-inch mesh—				
1..... per cent	98.50	97.50	98.00	97.50
2..... do	96.00	94.50	96.00	96.00
3..... do	95.00	93.00	94.50	94.50
4..... do	92.50	90.00	93.00	92.50
5..... do	90.50	89.50	92.50	94.00

Remarks.—Test 253: Light gray, with some little deposit of carbon. Breakage somewhat cross-fractured, but pieces drawn in large chunks. Cell structure large. Soft coke.

Test 255: Light gray and silvery, large deposit of carbon. Breakage good, large, long pieces. Cell structure a little large. Metallic ring. Good, heavy coke.

Tests 257 and 261: Same as test 255.

Chemical analyses.

Test No.	Laboratory No.		Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Phosphorus.
253	505-D...	Coal	Wet. 5.38	31.22	49.66	13.74	0.69
			Dry	33.00	52.48	14.52	.73
	509-D...	Coke	Wet. .43	.94	77.98	20.65	.51	0.0198
			Dry	.94	78.32	20.74	.51
255	510-D...	Coal	Wet. 8.38	32.28	48.98	10.36	.68
			Dry	35.23	53.47	11.30	.74
	515-D...	Coke	Wet. .37	1.42	80.73	17.48	.57	.0142
			Dry	1.43	81.03	17.54	.57
257	528-D...	Coal	Wet. 6.35	31.97	48.28	13.40	.65
			Dry	34.14	51.54	14.32	.69
	547-D...	Coke	Wet. .49	.04	78.43	21.04	.62	.0145
			Dry	.04	78.82	21.14	.62
261	541-D...	Coal	Wet. 6.53	31.05	48.42	14.00	.57
			Dry	33.23	51.80	14.97	.61
	549-D...	Coke	Wet. .56	.16	77.50	21.78	.52	.0179
			Dry	.16	77.94	21.90	.52

DENVER No. 18.

Coking tests.

	Test—				
	262.	263.	264.	265.	267.
Date.....	9, 22, 08	9, 23, 08	9, 23, 08	9, 25, 08	9, 27, 08
Duration.....hours..	63	33	39	34	54
Size:					
As shipped.....	r. o. m.	r. o. m.	r. o. m.	r. o. m.	r. o. m.
As used.....	w., f. c.	w., f. c.	w., f. c.	w., f. c.	w., f. c.
Coal charged:					
Wet.....pounds..	14,000	8,700	10,600	8,810	12,380
Dry.....do.....	13,548	7,806	9,799	8,063	11,944
Coke produced:					
Wet.....do.....	9,150	5,700	7,050	6,000	7,837
.....per cent..	65.35	65.52	66.51	68.10	63.30
Dry.....pounds..	9,144	5,667	7,017	5,998	7,824
.....per cent..	67.49	72.60	71.61	74.39	65.51
Breeze produced:					
Wet.....pounds..	264	175	249	197	300
.....per cent..	1.89	2.01	2.35	2.24	2.42
Dry.....pounds..	264	174	248	197	300
.....per cent..	1.95	2.23	2.53	2.44	2.51
Total yield:					
Wet.....do.....	67.24	67.53	68.86	70.34	65.72
Dry.....do.....	69.44	74.83	74.14	76.83	68.02
Physical properties of coke:					
Specific gravity—					
Apparent.....	1.11	1.03	1.10	1.08	1.08
Real.....	1.96	1.93	1.96	1.99	1.97
Volume—					
Coke.....per cent..	57.00	53.00	56.00	54.00	55.00
Cells.....do.....	43.00	47.00	44.00	46.00	45.00
Weight per cubic foot—					
Wet.....pounds..	96.01	98.15	95.71	96.01	95.33
Dry.....do.....	69.19	63.86	68.28	67.32	67.25
6-foot drop test over 2-inch mesh—					
1.....per cent..	98.00	98.50	98.00	97.50	98.50
2.....do.....	96.00	96.00	95.50	96.00	97.00
3.....do.....	94.50	94.50	94.00	94.50	95.00
4.....do.....	92.50	92.50	92.50	91.50	93.50
5.....do.....	94.00	93.50	94.50	94.00	95.50

Remarks.—Test 262: Light gray and silvery. Breakage good, long, large pieces. Cell structure small. Metallic ring. Good, strong, heavy coke.

Tests 263, 264, and 265: Same as 262.

Test 267: Light gray and silvery, large deposit of carbon. Breakage good, long, large, uniform-sized pieces. Cell structure small, dense. Metallic ring. Good, heavy, strong, hard coke.

Chemical analyses.

Test No.	Laboratory No.		Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Phosphorus.
262	543-D	Coal	Wet.. 3.23	31.82	55.41	9.54	0.48
			Dry..	32.88	57.26	9.86	.50
	558-D	Coke	Wet.. .07	.86	84.72	14.35	.65	0.1333
263	554-D	Coal	Wet.. 10.28	29.53	51.64	14.36	.65
			Dry..	32.91	57.56	9.53	.49
	559-D	Coke	Wet.. .58	1.65	84.02	13.75	.51	.1226
264	553-D	Coal	Wet.. 7.56	1.66	84.51	13.63	.51
			Dry..	29.24	52.00	11.20	.49
	560-D	Coke	Wet.. .47	31.63	56.26	12.11	.53
265	556-D	Coal	Wet.. 8.48	1.35	79.80	18.38	.48	.1631
			Dry..	1.35	80.19	18.46	.48
	557-D	Coke	Wet.. .03	28.51	51.43	11.58	.54
267	561-D	Coal	Wet.. 3.52	31.15	56.20	12.65	.59
			Dry..	2.26	78.54	19.17	.57	.1222
	568-D	Coke	Wet.. .16	2.26	78.56	19.18	.57
			Dry..	31.98	53.05	11.45	.50
				33.16	54.96	11.86	.52
				.88	82.31	16.65	.55	.1327
				.88	82.44	16.68	.55

DENVER No. 19.

Coking tests.

	Test—				
	269 (19B).	270 (19B).	272 (19A).	273 (19A).	274 (19C).
Date.....	9, 30, 08	10, 3, 08	10, 5, 08	10, 7, 08	10, 12, 08
Duration.....hours..	24	36	42	41	33
Size:					
As shipped.....	r. o. m. No. 5	r. o. m. No. 5	r. o. m. No. 5	r. o. m. No. 5	r. o. m. No. 4
As used.....	a w., f. c.	a w., f. c.	w., f. c.	w., f. c.	w., f. c.
Coal charged:					
Wet.....pounds..	12, 700	6, 330	8, 850	8, 020	6, 960
Dry.....do.....	12, 216	5, 984	7, 582	7, 189	6, 018
Coke produced:					
Wet.....do.....	None.	None.	3, 450	None.	None.
Dry.....pounds..			38.98		
Wet.....per cent.			3.448		
Dry.....pounds..			45.48		
Breeze produced:					
Wet.....pounds..			1, 050		
Dry.....per cent.			11.87		
Wet.....pounds..			1, 049		
Dry.....per cent.			13.84		
Total yield:					
Wet.....do.....			50.85		
Dry.....do.....			59.32		
Physical properties of coke:					
Specific gravity—					
Apparent.....			1.06		
Real.....			1.84		
Volume—					
Coke.....per cent.			58.00		
Cells.....do.....			42.00		
Weight per cubic foot—					
Wet.....pounds..			92.28		
Dry.....do.....			66.07		

a Washed at mine.

Remarks.—Tests 269, 270, 273, and 274 produced no coke.

Test 272: The upper 3 inches of the charge did not stick together or show signs of coking. The lower 12 inches fused and formed very soft coke, with slight cohesion and scarcely any definite cell structure. Breakage of charge drawn, very poor—large percentage went to breeze on dumping from barrows. No drop test was made.

Chemical analyses.

Test No.	Laboratory No.			Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Phosphorus.
269 (19B)	584-D...	Coal	Wet...	3.81	36.60	41.45	18.14	0.71
			Dry...	38.05	43.09	18.86	.74
270 (19B)	587-D...	Coal	Wet...	5.47	35.30	41.44	17.79	.67
			Dry...	37.35	43.83	18.82	.71
272 (19A)	591-D...	Coal	Wet...	14.33	30.61	40.24	14.82	.71
			Dry...	35.72	46.98	17.30	.83
272 (19A)	592-D...	Coke	Wet...	.05	2.78	76.42	20.75	.53	0.0550
			Dry...	2.78	76.46	20.76	.53
273 (19A)	602-D...	Coal	Wet...	10.36	33.20	41.98	14.46	.82
			Dry...	37.03	46.84	16.13	.92
274 (19C)	629-D...	Coal	Wet...	13.41	31.66	41.59	13.34	.61
			Dry...	36.56	48.04	15.40	.70

DENVER No. 20.

Coking tests.

	Test—			
	287.	288.	289.	290.
Date.....	11, 7, 08	11, 9, 08	11, 10, 08	11, 12, 08
Duration..... hours.	38	44	72	44
Size:				
As shipped.....	r. o. m.	r. o. m.	r. o. m.	r. o. m.
As used.....	w., n. c.	w., f. c.	w., f. c.	w., f. c.
Coal charged:				
Wet..... pounds.	10,580	12,400	16,300	12,300
Dry..... do.	9,607	11,463	15,139	11,643
Coke produced:				
Wet..... do.	6,654	7,950	10,350	8,100
..... per cent.	62.89	64.11	63.50	65.85
Dry..... pounds.	6,550	7,938	10,305	8,098
..... per cent.	68.18	69.25	68.07	69.55
Breeze produced:				
Wet..... pounds.	277	304	378	296
..... per cent.	2.62	2.45	2.32	2.40
Dry..... pounds.	273	304	376	296
..... per cent.	2.84	2.65	2.48	2.53
Total yield:				
Wet..... do.	65.51	66.56	66.82	66.25
Dry..... do.	71.02	71.90	70.55	72.08
Physical properties of coke:				
Specific gravity—				
Apparent.....	1.03	1.10	1.09	1.04
Real.....	1.95	1.99	1.97	1.97
Volume—				
Coke..... per cent.	53.00	55.00	55.00	53.00
Cells..... do.	47.00	45.00	45.00	47.00
Weight per cubic foot—				
Wet..... pounds.	92.54	96.55	95.75	94.11
Dry..... do.	63.25	68.47	67.67	64.81
6-foot drop test over 2-inch mesh—				
1..... per cent.	97.50	98.50	97.00	98.00
2..... do.	95.00	98.00	94.00	96.00
3..... do.	93.50	96.50	92.50	93.50
4..... do.	91.00	95.00	88.00	91.00
5..... do.	93.50	97.50	93.00	92.50

Remarks.—Tests 287 and 288: Light gray and silvery. Breakage good. Cell structure a little small. Good, strong, heavy coke.

Tests 289 and 290: Light gray and silvery, large deposit of carbon. Breakage good; long, large pieces. Cell structure small, dense. Metallic ring. Good, hard, strong, heavy coke.

Chemical analyses.

Test No.	Laboratory No.		Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Phosphorus.
287	786-D	Coal	(Wet.. 9.20	29.12	49.86	11.82	0.51
			(Dry..	32.07	54.91	13.02	.56
288	790-D	Coke	(Wet.. 1.57	2.03	78.11	18.29	.57	0.0460
			(Dry..	2.06	79.39	18.55	.58
289	788-D	Coal	(Wet.. 7.56	30.32	47.61	14.51	.56
			(Dry..	32.79	51.51	15.70	.61
290	794-D	Coke	(Wet.. .15	.16	78.51	20.18	.60	.0477
			(Dry..	.16	79.63	20.21	.60
291	791-D	Coal	(Wet.. 7.12	29.12	50.94	12.82	.53
			(Dry..	31.36	54.84	13.80	.57
292	799-D	Coke	(Wet.. .43	2.81	76.80	19.96	.55	.0579
			(Dry..	2.82	77.14	20.04	.55
293	795-D	Coal	(Wet.. 5.34	30.93	52.57	11.16	.53
			(Dry..	32.68	55.52	11.80	.56
294	801-D	Coke	(Wet.. .03	.71	80.99	18.27	.44	.0481
			(Dry..	.71	81.02	18.27	.44

DENVER No. 21.

Coking tests.

	Test—				
	277 (21A).	279 (21A).	278 (21B).	280 (21B).	281 (21B).
Date.....	10, 22, 08	10, 24, 08	10, 23, 08	10, 26, 08	10, 27, 08
Duration..... hours..	36	47	43	44	51
Size.....					
As shipped.....	r. o. m.	r. o. m.	1-in.	1-in.	1-in.
As used.....	r., f. c.	w., f. c.	r., f. c.	w., f. c.	w., f. c.
Coal charged:					
Wet..... pounds..	9,270	11,700	11,000	12,300	11,100
Dry..... do.....	8,572	10,815	10,745	11,646	10,109
Coke produced:					
Wet..... f...do....	5,670	6,996	6,450	7,200	6,309
per cent.....	61.17	59.79	58.64	58.54	56.84
Dry..... pounds..	5,649	6,951	6,407	7,129	6,158
per cent.....	65.90	64.27	59.63	61.21	60.92
Breeze produced:					
Wet..... pounds..	342	320	315	300	283
per cent.....	3.69	2.74	2.86	2.44	2.55
Dry..... pounds..	341	318	313	297	276
per cent.....	3.98	2.94	2.91	2.55	2.73
Total yield:					
Wet..... do.....	64.86	62.53	61.50	60.98	59.39
Dry..... do.....	69.88	67.21	63.54	63.76	63.65
Physical properties of coke:					
Specific gravity—					
Apparent.....	0.96	0.92	0.97	0.99	0.89
Real.....	1.94	1.90	1.94	1.91	1.89
Volume—					
Coke..... per cent..	49.00	48.00	50.00	47.00	47.00
Cells..... do.....	51.00	52.00	50.00	53.00	53.00
Weight per cubic foot—					
Wet..... pounds..	91.48	89.42	91.29	88.05	87.25
Dry..... do.....	59.66	57.00	60.08	54.98	54.18
6-foot drop test over 2-inch mesh—					
1..... per cent..	91.00	94.50	93.00	91.00	92.50
2..... do.....	83.50	82.00	83.50	84.50	87.00
3..... do.....	77.00	77.00	80.00	79.00	80.00
4..... do.....	71.00	70.50	73.00	73.50	74.50
5..... do.....	73.00	71.50	70.50	75.00	75.50

Remarks.—Test 277: Light gray and silvery, large deposit of carbon. Breakage, fingered. Very brittle. Top cemented together with deposited carbon. Cell structure a little large. Metallic ring.

Test 279: Same as from raw coal (test 277), except ash and sulphur lower.

Test 278: Same as test 277; not so much deposited carbon.

Test 280: Light gray and silvery. Breakage, long, fine-fingered pieces. Cell structure large; small at top and increasing gradually almost to sponge at bottom. Metallic ring. Brittle.

Test 281: Light gray and silvery. Breakage long, thin, fine-fingered pieces. Brittle. Cell structure a little large, but good, no sponge; slow draft probably the cause. Metallic ring. Good coke.

Chemical analyses.

Test No.	Laboratory No.		Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Phosphorus.
277 (21A)	692-D...	Coal.....	Wet..... 7.53	31.49	46.11	14.87	0.76
		Dry.....	34.06	49.86	15.08	.82
	696-D...	Coke.....	Wet..... .39	1.96	73.58	24.07	.53	0.1124
		Dry.....	1.97	73.86	24.17	.53
279 (21A)	695-D...	Coal.....	Wet..... 7.56	33.61	49.20	9.63	.68
		Dry.....	36.35	53.23	10.42	.74
	697-D...	Coke.....	Wet..... .64	1.30	81.03	17.03	.66	.1188
		Dry.....	1.31	81.65	17.14	.66
278 (21B)	694-D...	Coal.....	Wet..... 2.32	37.02	47.08	13.58	.60
		Dry.....	37.90	48.20	13.90	.61
	699-D...	Coke.....	Wet..... .67	.69	76.92	21.72	.38	.1137
		Dry.....70	77.43	21.87	.38
280 (21B)	698-D...	Coal.....	Wet..... 5.32	36.37	48.40	9.91	.63
		Dry.....	38.42	51.11	10.47	.67
	702-D...	Coke.....	Wet..... .99	2.42	70.42	26.17	.52	.1217
		Dry.....	2.44	71.13	26.43	.53
281 (21B)	700-D...	Coal.....	Wet..... 8.93	34.77	46.07	10.23	.56
		Dry.....	38.18	50.59	11.23	.62
	704-D...	Coke.....	Wet..... 2.40	1.83	79.45	16.32	.57	.1208
		Dry.....	1.88	81.40	16.72	.58

DENVER No. 22.

Coking tests.

	Test—	
	275.	276.
Date.....	10, 16, 08	10, 19, 08
Duration..... hours..	46	34
Size:		
As shipped.....	r. o. m.	r. o. m.
As used.....	w., f. c.	w., n. c.
Coal charged:		
Wet..... pounds..	10, 590	10, 200
Dry..... do.....	9, 245	8, 402
Coke produced:		
Wet..... f.do.....	6, 300	5, 930
..... per cent.....	59.49	58.13
Dry..... pounds..	6, 226	5, 865
..... per cent.....	67.45	69.80
Breeze produced:		
Wet..... pounds..	417	275
..... per cent.....	3.94	2.70
Dry..... pounds..	413	272
..... per cent.....	4.47	3.24
Total yield:		
Wet..... do.....	63.43	60.83
Dry..... do.....	71.92	73.04
Physical properties of coke:		
Specific gravity—		
Apparent.....	1.28	1.13
Real.....	2.01	1.91
Volume—		
Coke..... per cent..	64.00	59.00
Cells..... do.....	36.00	41.00
Weight per cubic foot—		
Wet..... pounds..	101.46	95.29
Dry..... do.....	79.02	69.72
6-foot drop test over 2-inch mesh—		
1..... per cent..	98.00	97.00
2..... do.....	97.00	94.00
3..... do.....	96.00	89.50
4..... do.....	94.50	85.00
5..... do.....	94.50	89.50

Remarks.—Test 275: Dull gray color. Breakage, large, irregular chunks. Cell structure very small. Soft, dense, tough coke. High ash and sulphur.

Test 276: Dull gray color. Breakage, large, irregular chunks. Cell structure small, but not so dense as in test 275. Heavy, dense coke. Ash and sulphur reduced by washing, ash reduction particularly noticeable.

Chemical analyses.

Test No.	Laboratory No.		Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Phosphorus.
275	{ 654-D ..	Coal	Wet.. 12.70	21.04	39.84	26.42	3.64
		Dry	24.10	45.64	30.26	4.17
	{ 690-D ..	Coke	Wet.. 1.02	1.12	61.48	36.37	3.19	0.0328
		Dry	1.14	62.12	36.74	3.22
276	{ 689-D ..	Coal	Wet.. 17.63	22.78	47.57	12.02	2.79
		Dry	27.65	57.76	14.89	3.39
	{ 691-D ..	Coke	Wet.. 1.10	.92	77.51	20.47	2.41	.0225
		Dry93	78.38	20.69	2.44

DENVER No. 23.

Coking tests.

	Test—				
	282.	283.	284.	285.	286.
Date	10, 28, 08	10, 31, 08	10, 31, 08	11, 3, 08	11, 5, 08
Duration	42	61	67	73	43
Size:					
As shipped	r. o. m.	r. o. m.	r. o. m.	r. o. m.	r. o. m.
As used	w., f. c.	w., f. c.	w., f. c.	w., f. c.	w., f. c.
Coal charged:					
Wet	9, 200	13, 150	14, 600	14, 660	11, 300
Dry	8, 681	11, 927	13, 311	12, 674	10, 249
Coke produced:					
Wet	5, 250	7, 180	7, 800	7, 612	7, 050
Dry	57.07	54.37	53.42	51.92	52.39
per cent..	5, 243	7, 119	7, 745	7, 603	7, 030
per cent..	60.40	59.69	58.19	59.99	68.59
Breeze produced:					
Wet	208	295	325	300	278
Dry	2.26	2.24	2.28	2.06	2.46
per cent..	208	294	323	300	277
per cent..	2.40	2.47	2.43	2.37	2.70
Total yield:					
Wet	59.33	56.61	55.65	53.97	64.85
Dry	62.80	62.16	60.62	62.36	71.28
Physical properties of coke:					
Specific gravity—					
Apparent	0.85	0.82	0.87	0.88	0.90
Real	1.91	1.87	1.91	1.91	1.99
Volume:					
Coke	45.00	44.00	46.00	46.00	45.00
Cells	55.00	56.00	54.00	54.00	55.00
Weight per cubic foot—					
Wet	87.21	85.84	87.55	88.51	90.26
Dry	82.92	80.90	83.87	84.83	85.97
6-foot drop test over 2-inch mesh					
1.	97.00	97.00	96.50	97.50	97.00
2.	94.50	94.50	94.00	94.00	95.00
3.	92.00	92.00	91.50	91.50	93.50
4.	89.50	89.50	90.00	89.50	92.00
5.	93.00	94.00	98.00	92.50	93.00

Remarks.—Tests 282, 283, 284, 285: Light gray and silvery. Breakage somewhat cross-fractured, but pieces of good, large, uniform size. Cell structure a little large. Metallic ring. Low percentage of yield probably due to burning of coke at door. High sulphur.

Test 286: Light gray and silvery. Breakage somewhat cross-fractured, not so good as four preceding tests, but pieces still of good, large, uniform size. Cell structure

a little large. Breakage and cells, as well as higher percentage of breeze, probably due to not crushing. High sulphur. Percentage of sulphur too high for iron blast furnace purposes, but should give good results in any practice where sulphur is not detrimental. Larger yield of coke due to small, slow draft.

Chemical analyses.

Test No.	Laboratory No.		Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Phosphorus.
282	708-D	Coal	Wet. 5.64	35.51	51.23	6.62	3.10
			Dry.....	38.69	54.29	7.02	3.29
	728-D	Coke	Wet. .13	.69	87.65	11.53	2.56	0.0399
			Dry.....	.69	87.77	11.54	2.56
283	706-D	Coal	Wet. 9.30	24.86	50.19	5.65	2.81
			Dry.....	38.44	55.33	6.23	3.10
	730-D	Coke	Wet. .44	.62	88.61	10.33	2.59	.0345
			Dry.....	.62	89.00	10.38	2.60
284	727-D	Coal	Wet. 8.83	24.32	50.96	5.89	3.13
			Dry.....	37.65	55.89	6.46	3.43
	731-D	Coke	Wet. .71	.15	88.68	10.46	2.75	.0327
			Dry.....	.15	89.32	10.53	2.77
285	729-D	Coal	Wet. 12.55	32.27	45.78	8.40	3.59
			Dry.....	37.33	52.95	9.72	4.15
	786-D	Coke	Wet. .12	2.85	82.90	14.23	3.44	.0380
			Dry.....	2.85	82.90	14.25	3.44
286	732-D	Coal	Wet. 9.30	34.33	49.18	7.19	3.50
			Dry.....	37.85	54.22	7.93	3.86
	789-D	Coke	Wet. .28	.12	87.38	12.22	2.67	.0354
			Dry.....	.12	87.63	12.25	2.68

DENVER No. 25.

Coking tests.

	Test—			
	291.	292.	295.	296.
Date.....	11, 14, 08	11, 16, 08	11, 23, 08	11, 27, 08
Duration..... hours..	40	45	62	87
Size:				
As shipped.....	1-in. scr.	1-in. scr.	1-in. scr.	1-in. scr.
As used.....	r., f. c.	w., f. c.	w., f. c.	w., f. c.
Coal charged:				
Wet..... pounds..	10,300	12,600	13,430	14,850
Dry..... do.....	9,897	11,800	12,467	13,600
Coke produced:				
Wet..... do.....	5,925	7,538	7,870	8,436
..... per cent..	57.52	59.40	58.60	56.81
Dry..... pounds..	5,791	7,518	7,776	8,353
..... per cent..	58.51	63.39	62.37	61.42
Breeze produced:				
Wet..... pounds..	525	683	600	684
..... per cent..	5.10	5.38	4.47	4.61
Dry..... pounds..	513	681	598	677
..... per cent..	5.18	5.74	4.76	4.98
Total yield:				
Wet..... do.....	62.62	64.78	68.07	61.42
Dry..... do.....	63.69	69.13	67.13	65.40
Physical properties of coke:				
Specific gravity—				
Apparent.....	1.01	0.98	0.97	1.01
Real.....	1.84	1.86	1.84	1.84
Volume—				
Coke..... per cent..	55.00	53.00	53.00	55.00
Cells..... do.....	45.00	47.00	47.00	45.00
Weight per cubic foot—				
Wet..... pounds..	89.05	90.26	89.05	89.45
Dry..... do.....	61.67	60.96	59.87	62.37
6-foot drop test over 2-inch mesh—				
1..... per cent..	97.50	94.50	97.90	94.50
2..... do.....	95.00	88.00	91.50	89.50
3..... do.....	92.50	81.00	86.00	84.00
4..... do.....	91.00	78.00	82.00	79.50
5..... do.....	93.80	81.50	88.00	83.00

Remarks.—Test 291: Light gray, some little silvery deposit of carbon. Breakage, good, long pieces, somewhat fingered. Cell structure small.

Test 292: Light gray, some little silvery deposit of carbon. Breakage, two distinct layers, practically dividing charge in two parts, drawn in 9-inch chunks similar to by-product coke. Cell structure small. Metallic ring. Very brittle. Ash reduced by washing.

Test 295: Light gray and silvery. Breakage same as in test 292, except that chunks were 12 and 6 inches in layers, pieces from each layer nearly uniform in size and breaking but little after drawn. Cell structure very small and dense. Good, heavy, dense coke, but brittle.

Test 296: Light gray and silvery. Two layers of 16 and 8 inches, the upper 16 inches somewhat fingered, the lower 8 inches in 8-inch cubes of uniform size. Cell structure very small and dense. Good, heavy, dense, coke, but brittle.

Chemical analyses.

Test No.	Laboratory No.		Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Phosphorus.
291	796-D...	Coal.....	Wet..	3.91	33.85	47.28	14.96	0.75
			Dry..		35.24	46.19	15.57	.78
	802-D...	Coke.....	Wet..	2.26	1.79	77.40	18.55	.59
			Dry..		1.83	79.19	18.98	.60
292	800-D...	Coal.....	Wet..	6.54	26.51	48.31	8.64	.70
			Dry..		39.06	51.70	9.24	.75
	806-D...	Coke.....	Wet..	.27	.34	84.12	15.27	.47
			Dry..		.34	84.36	15.30	.47
295	837-D...	Coal.....	Wet..	7.17	34.97	48.62	9.24	.66
			Dry..		37.68	52.37	9.95	.71
	842-D...	Coke.....	Wet..	1.20	1.94	81.76	15.10	.49
			Dry..		1.96	82.75	15.29	.50
296	841-D...	Coal.....	Wet..	8.42	33.18	48.37	10.03	.69
			Dry..		36.23	52.82	10.95	.75
	856-D...	Coke.....	Wet..	.98	1.14	82.22	15.65	.65
			Dry..		1.15	83.04	15.81	.66

DENVER No. 26.

Coking tests.

		Test 293.
Date.....		11, 18, 08
Duration.....	hours..	48
Size:		
As shipped.....		r. o. m.
As used.....		r., f. c.
Coal charged (wet).....	pounds..	9,300
Coke produced.....		None.

Remarks.—Test 293: No coke produced. All volatile apparently expelled. Drawn from oven in same condition as charged, each separate piece coming out in size and shape as charged. No evidence of fusion or coking. Heavy clinker over whole top of charge. This is subbituminous coal.

Chemical analyses.

Test No.	Laboratory No.		Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Phosphorus.
293	803-D...	Coal.....	{Wet... 19.28 Dry... 42.87	34.61 42.87	41.41 51.30	4.70 5.83	0.39 .48

DENVER No. 27.*Coking tests.*

	Test—			
	297.	300.	301.	302.
Date.....	11, 28, 08	12, 5, 08	12, 8, 08	12, 7, 08
Duration..... hours..	36	57	65	48
Size:				
As shipped.....	r. o. m.	r. o. m.	r. o. m.	r. o. m.
As used.....	r., f. c.	r., f. c.	w., f. c.	w., f. c.
Coal charged:				
Wet..... pounds..	10,650	13,080	14,400	11,100
Dry..... do.....	10,315	12,668	13,668	10,645
Coke produced:				
Wet..... /do.....	7,200	8,700	9,150	7,350
..... per cent..	67.61	66.51	63.54	66.22
Dry..... pounds..	7,184	8,661	9,125	7,327
..... per cent..	69.66	68.37	66.76	68.83
Breeze produced:				
Wet..... /pounds..	274	300	373	219
..... per cent..	2.57	2.29	2.59	1.97
Dry..... pounds..	273	299	372	218
..... per cent..	2.65	2.36	2.72	2.05
Total yield:				
Wet..... do.....	70.18	68.80	66.13	66.19
Dry..... do.....	72.30	70.73	69.48	70.88
Physical properties of coke:				
Specific gravity—				
Apparent.....	0.93	0.93	0.88	0.85
Real.....	2.04	2.10	2.09	2.11
Volume—				
Coke..... per cent..	46.00	42.00	42.00	40.00
Cells..... do.....	54.00	58.00	58.00	60.00
Weight per cubic foot—				
Wet..... pounds..	91.55	93.88	90.91	90.26
Dry..... do.....	87.87	87.72	84.75	82.84
6-foot drop test over 2-inch mesh—				
1..... per cent..	97.50	96.50	97.50	97.50
2..... do.....	94.00	94.00	95.50	95.00
3..... do.....	91.00	92.00	93.00	92.50
4..... do.....	87.50	87.00	91.50	89.50
5..... do.....	90.50	92.50	94.00	93.50

Remarks.—Test 297: Light gray, some little silvery deposit of carbon and 2-inch black butts. Breakage badly cross fractured. Cell structure large. Metallic ring. Washing, larger charge, and slow draft probably would improve quality.

Test 300: Light gray and silvery, butts removed. Breakage badly cross fractured, but good. Cell structure large. Metallic ring. One inch less draft.

Test 301: Light gray and silvery. Breakage badly cross fractured, but pieces of good, large, uniform size. Cell structure large. Metallic ring.

Test 302: Light gray, some little silvery deposit of carbon. Breakage, badly cross fractured, but pieces of good, large, uniform size. Cell structure large. Metallic ring.

Chemical analyses.

Test No.	Laboratory No.		Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Phosphorus.
297	851-D...	Coal.....	Wet.. 3.15	33.39	49.65	13.81	0.66
			Dry.....	34.47	51.27	14.26	.68
	854-D...	Coke.....	Wet.. .22	.73	79.96	19.09	.55	0.0113
			Dry.....	.73	80.13	19.14	.56
300	858-D...	Coal.....	Wet.. 3.15	34.42	47.71	14.72	.58
			Dry.....	35.55	49.25	15.20	.60
	885-D...	Coke.....	Wet.. .45	1.11	78.27	20.17	.49	.0105
			Dry.....	1.12	78.62	20.26	.49
301	886-D...	Coal.....	Wet.. 5.08	32.00	52.78	10.14	.59
			Dry.....	33.71	55.61	10.68	.62
	896-D...	Coke.....	Wet.. .27	.55	84.37	14.80	.54	.0114
			Dry.....	.55	84.60	14.84	.54
302	887-D...	Coal.....	Wet.. 4.10	31.66	54.16	10.08	.61
			Dry.....	33.01	56.48	10.51	.64
	900-D...	Coke.....	Wet.. .31	.49	85.01	14.19	.49	.0109
			Dry.....	.50	85.27	14.23	.50

DENVER No. 28.*Coking tests.*

		Test—		
		298.	303.	305.
Date.....		11, 28, 08	12, 9, 08	12, 11, 08
Duration.....	hours..	24	39	36
Size:				
As shipped.....		r. o. m.	r. o. m.	r. o. m.
As used.....		r., f. c.	w., f. c.	w., n. c.
Coal charged:				
Wet.....	pounds..	12, 930	10, 510	6, 510
Dry.....	do.....	12, 049	9, 549	5, 959
Coke produced.....		None.	None.	None.

Remarks.—Test 298: No coke produced. Burned for twenty-four hours with very hot flame. Did not ash.

Test 303: No coke produced. All volatile expelled. Drawn from oven in same size as charged. No evidence of fusion or coherence.

Test 305: This was an attempt to produce coke from coal not crushed, with the oven heated up with a charge of red-hot coke at the door. The product showed some signs of coking. High heat of by-product ovens might give better results.

Chemical analyses.

Test No.	Laboratory No.		Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.
298	853-D....	Coal.....	Wet.. 6.81	35.28	46.50	11.41	0.60
			Dry.....	37.86	49.89	12.25	.64
303	898-D....	Coal.....	Wet.. 9.28	34.06	48.79	7.92	.55
			Dry.....	37.52	53.75	8.73	.60
305	892-D....	Coal.....	Wet.. 8.46	32.01	50.83	8.70	.57
			Dry.....	34.96	55.34	9.50	.62

DENVER No. 29.

Coking tests.

		Test—	
		299.	306.
Date.....		12, 2, 08	12, 15, 08
Duration.....	hours..	77	82
Size:			
As shipped.....		r. o. m.	r. o. m.
As used.....		r., f. c.	w., f. c.
Coal charged:			
Wet.....	pounds..	13, 000	13, 450
Dry.....	do.	12, 757	12, 815
Coke produced:			
Wet.....	(do.)	8, 999	9, 150
	(per cent.)	68. 45	68. 03
Dry.....	(pounds.)	8, 853	9, 116
	(per cent.)	69. 40	71. 13
Breeze produced:			
Wet.....	(pounds.)	258	218
	(per cent.)	1. 99	1. 62
Dry.....	(pounds.)	257	217
	(per cent.)	2. 01	1. 69
Total yield:			
Wet.....	do.	70. 44	69. 65
Dry.....	do.	71. 41	72. 82
Physical properties of coke:			
Specific gravity—			
Apparent.....		0. 80	0. 79
Real.....		1. 99	2. 06
Volume—			
Coke.....	per cent..	40. 00	38. 00
Cells.....	do.	60. 00	62. 00
Weight per cubic foot—			
Wet.....	pounds..	87. 06	87. 74
Dry.....	do.	49. 64	49. 07
6-foot drop test over 2-inch mesh—			
1.....	per cent..	99. 00	99. 00
2.....	do.	98. 00	97. 50
3.....	do.	96. 50	97. 00
4.....	do.	95. 50	96. 00
5.....	do.	97. 00	97. 50

Remarks.—Test 299: Dull gray color, some little deposit of carbon. Breakage, very badly cross fractured, but pieces of good size. Not brittle, very little breaking down to breeze. Cell structure very small, dense. Percentage of cells by volume very large, but cells very small.

Test 306: Same as test 299. Ash reduced by washing.

Chemical analyses.

Test No.	Laboratory No.		Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Phosphorus.
299	855-D...	Coal.....	Wet.. 1. 87	22. 99	67. 08	8. 16	0. 70
			Dry.. .	23. 32	68. 36	8. 32	. 71
	881-D...	Coke.....	Wet.. . 52	1. 41	87. 77	10. 30	. 48	0. 0404
			Dry.. .	1. 42	88. 23	10. 35	. 48
306	897-D...	Coal.....	Wet.. 4. 72	22. 34	66. 85	6. 19	. 50
			Dry.. .	23. 34	70. 17	6. 49	. 52
	917-D...	Coke.....	Wet.. . 37	. 26	91. 33	8. 02	. 58	. 0425
			Dry.. .	. 26	91. 67	8. 05	. 58

DENVER No. 80.

Coking tests.

		Test 304.
Date.....		12, 12, 08
Duration.....	hours..	67
Size:		
As shipped.....		r. o. m.
As used.....		w., f. o.
Coal charged:		
Wet.....	pounds..	13, 110
Dry.....	do.	12, 313
Coke produced:		
Wet.....	{ do.	8, 850
	{ per cent..	67. 51
Dry.....	{ pounds..	8, 860
	{ per cent..	71. 47
Breeze produced:		
Wet.....	{ pounds..	313
	{ per cent..	2. 38
Dry.....	{ pounds..	311
	{ per cent..	2. 52
Total yield:		
Wet.....	do.	66. 89
Dry.....	do.	73. 99
Physical properties of coke:		
Specific gravity—		
Apparent.....		0. 87
Real.....		2. 08
Volume—		
Coke.....	per cent..	42. 00
Cells.....	do.	58. 00
Weight per cubic foot—		
Wet.....	pounds..	90. 11
Dry.....	do.	53. 95
6-foot drop test over 2-inch mesh—		
1.....	per cent..	96. 50
2.....	do.	97. 00
3.....	do.	94. 50
4.....	do.	93. 00
5.....	do.	95. 50

Remarks.—Test 304: Dull gray color, some little deposit of carbon. Breakage, very badly cross fractured, but pieces of good uniform size and not brittle. Cell structure very small, dense. High yield of coke and low percentage of breeze. Drop tests show very small breakage.

Chemical analyses.

Test No.	Laboratory No.		Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Phosphorus.
304	891-D	Coal.....	{ Wet..	6. 08	28. 06	62. 12	8. 74	0. 43
			{ Dry..		24. 55	66. 14	9. 31	. 46
	890-D	Coke.....	{ Wet..	. 57	1. 96	85. 82	11. 65	. 48
			{ Dry..		1. 97	86. 31	11. 72	. 48

DENVER No. 81.

Coking tests.

	Test—		
	311.	312.	313.
Date.....	12, 28, 8	12, 31, 8	12, 30, 8
Duration..... hours..	37	64	37
Size:			
As shipped.....	r. o. m.	r. o. m.	r. o. m.
As used.....	r., f. c.	w., f. c.	w., f. c.
Coal charged:			
Wet..... pounds..	10,400	13,200	9,000
Dry..... do.....	10,097	12,726	8,580
Coke produced:			
Wet..... { do.....	6,150	8,100	5,100
{ per cent.....	59.13	61.86	56.67
Dry..... { pounds.....	6,071	8,055	4,967
{ per cent.....	60.13	63.30	57.99
Breeze produced:			
Wet..... { pounds.....	450	332	217
{ per cent.....	4.33	2.52	2.41
Dry..... { pounds.....	444	330	211
{ per cent.....	4.40	2.59	2.46
Total yield:			
Wet..... do.....	63.46	63.88	59.08
Dry..... do.....	64.53	65.89	60.35
Physical properties of coke:			
Specific gravity—			
Apparent.....	0.97	0.97	0.93
Real.....	2.16	2.14	2.21
Volume—			
Coke..... per cent..	45.00	45.00	42.00
Cells..... do.....	55.00	55.00	58.00
Weight per cubic foot—			
Wet..... pounds..	94.03	94.45	92.62
Dry..... do.....	59.74	60.16	56.46
6-foot drop test over 2-inch mesh—			
1..... per cent..	92.50	94.50	96.00
2..... do.....	82.00	89.00	89.50
3..... do.....	76.00	84.00	84.00
4..... do.....	66.00	80.50	80.50
5..... do.....	87.50	87.00	86.50

Remarks.—Test 311: Dull gray color. Breakage, badly cross fractured, fine fingered, breaking badly. Cell structure little large. Soft coke.

Test 312: Gray color, some little silvery deposit of carbon. Breakage, badly cross fractured, fine fingered, and brittle. Cell structure a little large. Metallic ring. Improvement over raw charge, as well as small washed charge.

Test 313: Dull gray color, some little carbon. Breakage, very badly cross fractured. Fingered, soft, and brittle. Cell structure a little large, but better than that from raw charge. Ash reduced by washing.

Chemical analyses.

Test No.	Laboratory No.		Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Phosphorus.
311	960-D...	Coal.....	Wet.. 2.91	36.78	46.92	13.39	0.90
			Dry..	37.86	48.53	13.79	.98
	976-D...	Coke.....	Wet.. 1.29	1.61	80.01	17.09	.73	0.0113
			Dry..	1.63	81.06	17.31	.74
312	974-D...	Coal.....	Wet.. 3.59	28.97	47.43	10.01	.81
			Dry..	40.42	49.20	10.38	.84
	986-D...	Coke.....	Wet.. .55	.60	83.29	15.60	.59	.0122
			Dry..	.60	83.76	15.65	.59
313	975-D...	Coal.....	Wet.. 4.67	39.50	45.49	9.84	.76
			Dry..	41.44	48.76	9.80	.80
	978-D...	Coke.....	Wet.. 2.60	1.55	81.76	14.09	.59	.0114
			Dry..	1.59	83.95	14.46	.61

DENVER No. 32.

Coking tests.

	Test—		
	308.	309.	310.
Date.....	12, 24, 8	12, 26, 8	12, 28, 8
Duration..... hours	41	36	45
Size:			
As shipped.....	r. o. m.	r. o. m.	r. o. m.
As used.....	r., f. c.	w., f. c.	w., f. c.
Coal charged:			
Wet..... pounds	10, 080	9, 470	8, 100
Dry..... do	9, 270	8, 584	7, 291
Coke produced.....	None.	None.	None.

Remarks.—Test 308: No coke produced. No evidence of fusion or coking when the oven was drawn after forty hours.

Tests 309 and 310: No coke produced. Volatile was practically all expelled, but coal was drawn from the oven the same size as charged. No evidence of fusion or coking. Charges ashed down about 1 inch over the whole oven.

Chemical analyses.

Test No.	Laboratory No.		Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.
308	967-D.....	Coal.....	Wet.. 8.04	36.93	45.61	9.42	1.63
			Dry.. 40.16	49.60	10.24	1.77	
309	968-D.....	Coal.....	Wet.. 9.36	36.97	45.90	7.77	1.40
			Dry.. 40.79	50.64	8.57	1.54	
310	970-D.....	Coal.....	Wet.. 9.99	36.84	46.17	7.00	1.38
			Dry.. 40.93	51.29	7.78	1.53	

DENVER No. 33.

Coking tests.

	Test—	
	314.	315.
Date.....	12, 31, 8	1, 2, 9
Duration..... hours	28	31
Size:		
As shipped.....	r. o. m.	r. o. m.
As used.....	r., f. c.	w., f. c.
Coal charged:		
Wet..... pounds	10, 600	10, 100
Dry..... do	9, 661	9, 204
Coke produced.....	None.	None.

Remarks.—Tests 314 and 315: Burned for 28 and 31 hours, respectively. No evidence of fusion or coking.

Chemical analyses.

Test No.	Laboratory No.		Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.
314	977-D.....	Coal.....	Wet.. 8.86	37.96	48.56	4.62	1.00
			Dry.. 41.65	53.28	5.07	1.10	
315	966-D.....	Coal.....	Wet.. 8.87	38.62	48.69	3.82	.82
			Dry.. 42.38	53.43	4.19	.90	

DENVER Nos. 19A AND 20 MIXED IN EQUAL PORTIONS.

Coking tests.

		Test 294.
Date.....		11, 19, 08
Duration.....	hours	27
Size:		
As shipped.....		r. o. m.
As used.....		w., l. c.
Coal charged:		
Wet.....	pounds	8,450
Dry.....	do	8,002
Coke produced:		
Wet.....	{...do	4,886
	{per cent.	57.52
Dry.....	{pounds	4,868
	{per cent.	60.77
Breeze produced:		
Wet.....	{pounds	475
	{per cent.	5.62
Dry.....	{pounds	473
	{per cent.	5.91
Total yield:		
Wet.....	do	62.44
Dry.....	do	66.68
Physical properties of coke:		
Specific gravity—		
Apparent.....		1.05
Real.....		1.94
Volume—		
Coke.....	per cent.	54.00
Cells.....	do	46.00
Weight per cubic foot—		
Wet.....	pounds	93.88
Dry.....	do	65.19
6-foot drop test over 2-inch mesh—		
1.....	per cent.	98.50
2.....	do	97.00
3.....	do	94.50
4.....	do	92.50
5.....	do	94.00

Remarks.—Test 294: Dull gray color. Breakage, good. Cell structure very small, dense. Poor, soft, punky coke. Addition of noncoking (19A) to good coking (20) coal changed otherwise good coke into one of poor quality.

Chemical analyses.

Test No.	Laboratory No.		Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Phosphorus.
294	807-D...	Coal.....	Wet..	5.30	33.49	45.78	15.43	0.62
			Dry..		35.36	48.35	16.29	
	830-D...	Coke.....	Wet..	.48	.91	74.81	23.80	.55
			Dry..		.91	75.17	23.92	.55

DENVER Nos. 29 AND 30.

Coking tests.

		Test 307.
Date.....		12, 16, 8
Duration.....	hours..	75
Size:		
As shipped.....		r. o. m.
As used.....		w., l. c.
Coal charged:		
Wet.....	pounds..	13, 650
Dry.....	do.	13, 096
Coke produced:		
Wet.....	(...do.	9, 150
	(per cent..	67.03
Dry.....	pounds..	9, 013
	(per cent..	68.82
Breeze produced:		
Wet.....	(pounds..	226
	(per cent..	1.66
Dry.....	(pounds..	223
	(per cent..	1.70
Total yield:		
Wet.....	do.	68.69
Dry.....	do.	70.55
Physical properties of coke:		
Specific gravity—		
Apparent.....		0.84
Real.....		2.11
Volume—		
Coke.....	per cent..	40.00
Cells.....	do.	80.00
Weight per cubic foot—		
Wet.....	pounds..	89.00
Dry.....	do.	81.59
6-foot drop test over 2-inch mesh—		
1.....	per cent..	98.00
2.....	do.	97.50
3.....	do.	96.50
4.....	do.	95.50
5.....	do.	97.50

Remarks.—Test 307: These two coals were from the same bed, but only the upper coal (Denver No. 29) is used (raw) for coking. The object of this test was to show the possibilities of mixing and washing and then coking. The result warrants use of these coals if washed. Coke produced of dull gray color, very badly cross fractured, but of uniform size. Metallic ring. Very small cell structure, dense. Noticeable for small percentage of breeze produced and high yield of coke. Results practically the same as for the separate coals, tests 306 and 304.

Chemical analyses.

Test No.	Laboratory No.		Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Phosphorus.
307	898-D...	Coal.....	(Wet.. 4.06	22.82	65.47	7.65	0.46
			(Dry.. 23.79	68.23	7.98	.48
	918-D...	Coke.....	(Wet.. 1.50	1.26	87.50	9.74	.61	0.0358
			(Dry.. 1.28	88.83	9.89	.62

NOTES ON TESTS AND CONCLUSIONS.

Denver No. 15, from Illinois, was tested with a view of producing metallurgical coke from coal from a field that had shown promise in previous tests at the St. Louis plant. The coal from a newly opened mine, not available during former tests, demonstrated by actual test in a copper smelter that good commercial metallurgical coke can be produced from Illinois coal in beehive ovens.

Denver No. 23, from the Weir-Pittsburg seam in Kansas, was tested to show the possibility of producing coke for zinc smelting, one of the State's large industries, for which coke is at present being imported from other States. The results show that coke of good quality, with low ash but high sulphur content, can be produced from some Kansas coal. The high sulphur of this coal, 5.38 per cent, is reduced 42 per cent by washing, and there is a further reduction of practically 18 per cent in conversion of coal to coke. The resultant sulphur, 2.56 per cent, though still high, is no detriment in zinc smelting.

Denver No. 19A, with addition of 50 per cent coking coal, produced soft, poor coke. The addition of noncoking coal to good coking coal changed the product from good coke to one of poor quality.

Denver No. 26 is subbituminous and possesses no coking qualities.

Denver Nos. 29 and 30 are from the same mine, the upper portion (29) being mined and used for manufacture of coke and the lower portion (30) being left in the mine. These two coals were mixed in equal portions, washed, and the resulting washed coal coked, producing about the same grade of coke as that from the separate portions. The results seemingly show that the whole seam of coal is available for the production of coke.

Denver No. 32 showed no sign of fusion or coking in the ovens, though commercial laboratories had reported it as coking coal.

The loss of sulphur in the 56 tests that produced coke averages 47.71 per cent, the lowest being 8.70 per cent for Denver Nos. 29 and 30 (test 307) and the highest 66.20 per cent for Denver No. 19A (test 272).

PUBLICATIONS ON FUEL TESTING.

The following publications, except those to which a price is affixed, can be obtained free by applying to the Director of the Bureau of Mines, Washington, D. C. The priced publications can be purchased from the Superintendent of Documents, Government Printing Office, Washington, D. C.

PUBLICATIONS OF THE UNITED STATES GEOLOGICAL SURVEY.

BULLETIN 261. Preliminary report on the operations of the coal-testing plant of the United States Geological Survey at the Louisiana Purchase Exposition, in St. Louis, Mo., 1904; E. W. Parker, J. A. Holmes, M. R. Campbell, committee in charge. 1905. 172 pp. 10 cents.

PROFESSIONAL PAPER 48. Report on the operations of the coal-testing plant of the United States Geological Survey at the Louisiana Purchase Exposition, St. Louis, Mo., 1904; E. W. Parker, J. A. Holmes, M. R. Campbell, committee in charge. 1906. In three parts. 1492 pp., 13 pls. \$1.50.

BULLETIN 290. Preliminary report on the operations of the fuel-testing plant of the United States Geological Survey at St. Louis, Mo., 1905, by J. A. Holmes. 1906. 240 pp. 20 cents.

BULLETIN 323. Experimental work conducted in the chemical laboratory of the United States fuel-testing plant at St. Louis, Mo., January 1, 1905, to July 31, 1906, by N. W. Lord. 1907. 49 pp. 10 cents.

BULLETIN 325. A study of four hundred steaming tests made at the fuel-testing plant, St. Louis, Mo., 1904, 1905, and 1906, by L. P. Breckenridge. 1907. 196 pp. 20 cents.

BULLETIN 332. Report of the United States fuel-testing plant at St. Louis, Mo., January 1, 1906, to June 30, 1907; J. A. Holmes, in charge. 1908. 299 pp. 25 cents.

BULLETIN 334. The burning of coal without smoke in boiler plants; a preliminary report, by D. T. Randall. 1908. 26 pp. 5 cents. (See Bull. 373.)

BULLETIN 336. Washing and coking tests of coal and cupola tests of coke, by Richard Moldenke, A. W. Belden, and G. R. Delamater. 1908. 76 pp. 10 cents.

BULLETIN 339. The purchase of coal under government and commercial specifications on the basis of its heating value, with analyses of coal delivered under government contracts, by D. T. Randall. 1908. 27 pp. 5 cents. (See Bull. 428.)

BULLETIN 343. Binders for coal briquets, by J. E. Mills. 1908. 56 pp.

BULLETIN 362. Mine sampling and chemical analyses of coals tested at the United States fuel-testing plant, Norfolk, Va., in 1907, by J. S. Burrows. 1908. 23 pp. 5 cents.

BULLETIN 363. Comparative tests of run-of-mine and briquetted coal on locomotives, including torpedo-boat tests and some foreign specifications for briquetted fuel, by W. F. M. Goss. 1908. 57 pp., 4 pls.

BULLETIN 366. Tests of coal and briquets as fuel for house-heating boilers, by D. T. Randall. 1908. 44 pp., 3 pls.

BULLETIN 367. Significance of drafts in steam-boiler practice, by W. T. Ray and Henry Kreisinger. 1909. 61 pp.

BULLETIN 368. Washing and coking tests of coal at Denver, Colo., by A. W. Belden, G. R. Delamater, and J. W. Groves. 1909. 54 pp., 2 pls.

BULLETIN 373. The smokeless combustion of coal in boiler plants, by D. T. Randall and H. W. Weeks. 1909. 188 pp. 20 cents.

BULLETIN 378. The purchase of coal under government specifications, by J. S. Burrows. 1909. 44 pp. 10 cents. (See Bull. 428.)

BULLETIN 382. The effect of oxygen in coal, by David White. 1909. 78 pp., 3 pls.

BULLETIN 385. Briquetting tests at the United States fuel-testing plant, Norfolk, Va., 1907-8, by C. L. Wright. 1909. 41 pp., 9 pls.

BULLETIN 392. Commercial deductions from comparisons of gasoline and alcohol tests on internal-combustion engines, by R. M. Strong. 1909. 38 pp.

BULLETIN 393. Incidental problems in gas-producer tests, by R. H. Fernald, C. D. Smith, J. K. Clement, and H. A. Grine. 1909. 29 pp.

BULLETIN 402. The utilization of fuel in locomotive practice, by W. F. M. Goss. 1909. 28 pp.

BULLETIN 403. Comparative tests of run-of-mine and briquetted coal on the torpedo boat *Biddle*, by Walter T. Ray and Henry Kreisinger. 1909. 49 pp.

BULLETIN 412. Tests of run-of-mine and briquetted coal in a locomotive boiler, by Walter T. Ray and Henry Kreisinger. 1909. 32 pp.

BULLETIN 416. Recent development of the producer-gas power plant in the United States, by R. H. Fernald. 1909. 82 pp., 2 pls. 15 cents.

BULLETIN 428. The purchase of coal by the Government under specifications, with analyses of coal delivered for the fiscal year 1908-9, by G. S. Pope. 80 pp. 10 cents.

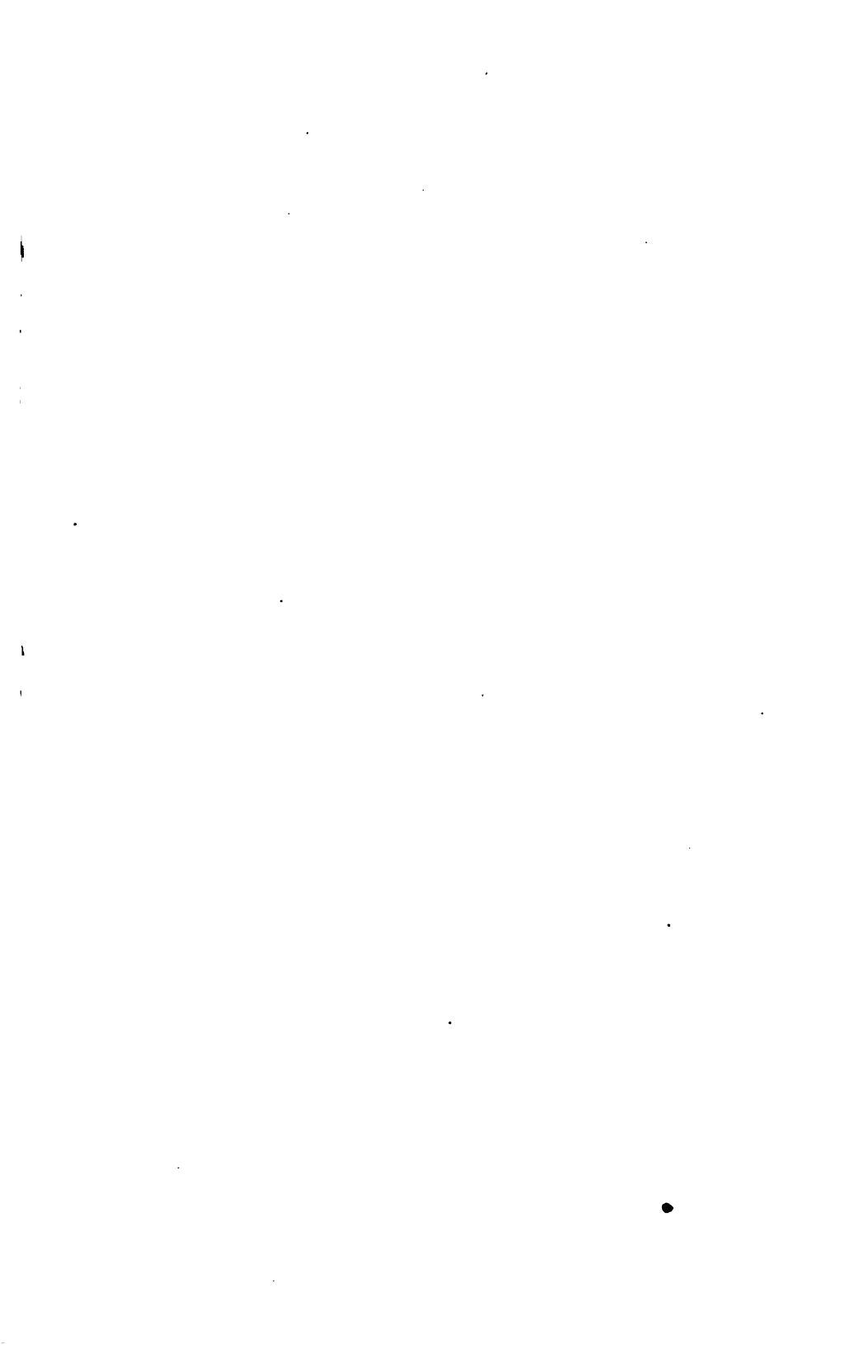
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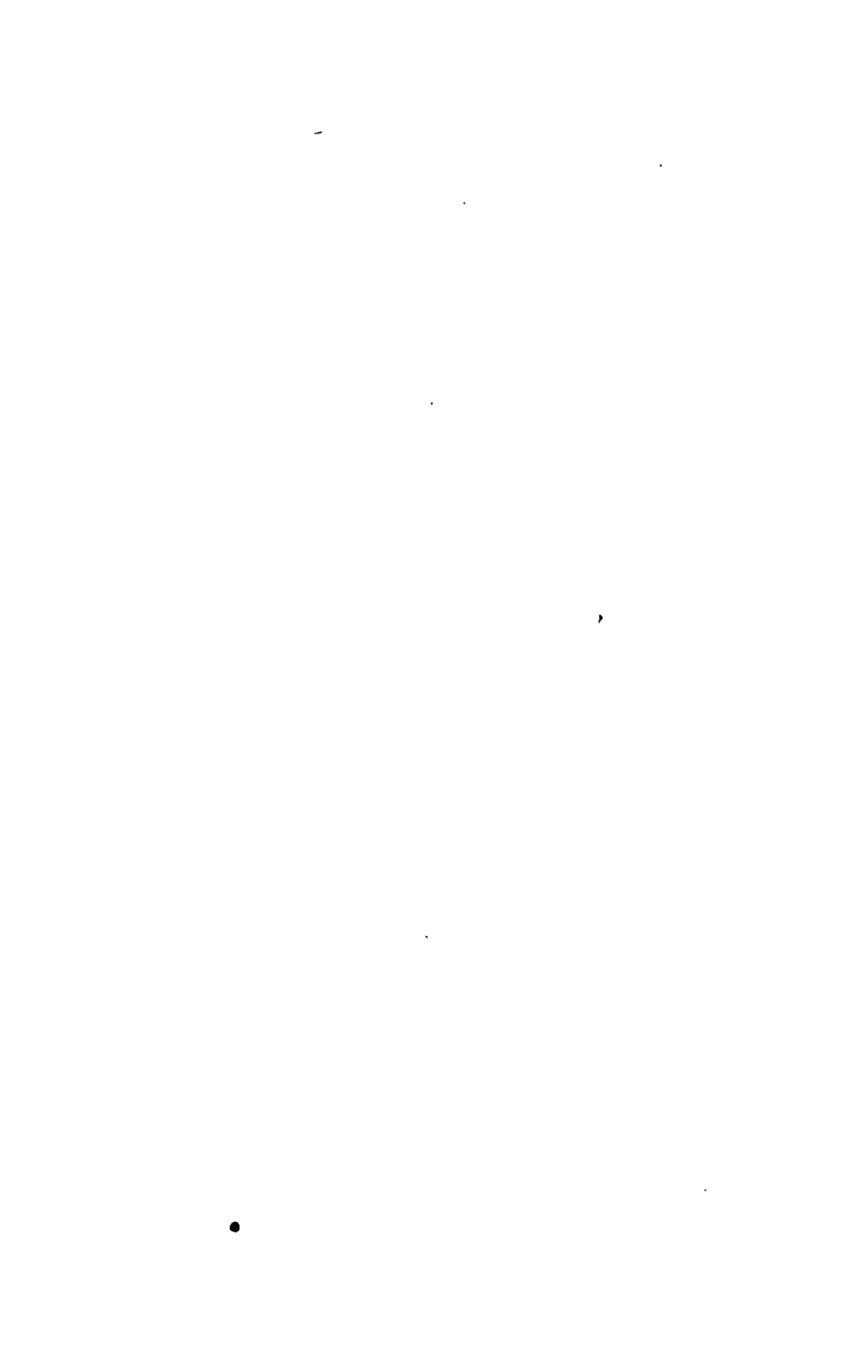
BULLETIN 1. The volatile matter of coal, by H. C. Porter and F. K. Ovitz. 1910. 56 pp., 1 pl.

BULLETIN 2. North Dakota lignite as a fuel for power-plant boilers, by D. T. Randall and Henry Kreisinger. 1910. 42 pp., 1 pl.

BULLETIN 3. The coke industry of the United States as related to the foundry, by Richard Moldenke. 1910. 32 pp.

BULLETIN 4. Features of producer-gas power-plant development in Europe, by R. H. Fernald. 1910.





Bulletin 6

DEPARTMENT OF THE INTERIOR

BUREAU OF MINES

JOSEPH A. HOLMES, DIRECTOR

COALS
AVAILABLE FOR THE MANUFACTURE OF
ILLUMINATING GAS

BY

A. H. WHITE AND PERRY BARKER

COMPILED AND REVISED BY

HERBERT M. WILSON



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COALS AVAILABLE FOR THE MANUFACTURE OF ILLUMINATING GAS.

By A. H. WHITE and PERRY BARKER.

INTRODUCTION.

By HERBERT M. WILSON.

GENERAL STATEMENT.

In a consideration of the various means whereby more economical and more efficient use may be made of the fuels in the United States, the possibility of obtaining for the production of illuminating gas other and cheaper fuels than the Pennsylvania coals demands attention. For the Government, as well as for private corporations and the householder, there can be no more economical and efficient way of using some coals than through the medium of illuminating gas. In the stove, gas reduces the labor cost of heat production and lessens the drudgery of the kitchen; burned in the Welsbach mantle, it is an excellent and cheap illuminant. In addition, the coke that remains after the gas has been recovered furnishes a smokeless fuel that has about the same heating value as anthracite. Hence any investigations that will indicate how local coals through proper treatment may be substituted for the higher priced and rapidly vanishing Pennsylvania gas coals will bring about lower prices for both gas and coke and will also aid to conserve for use in metallurgical processes the coking coals of Pennsylvania and of other States.

The annual drain on the gas-coal resources of this country and the importance of the gas and coke industries are indicated by the fact that 8,390,129 tons of coal were carbonized in retorts in the United States in 1909. The resulting salable products from by-product ovens were 15,791,220,000 cubic feet of coal gas, 6,254,644 tons of coke, and 60,126,000 gallons of tar. The total value of all by-products was about \$28,508,637.

There are few well-developed coal fields in this country that furnish coal satisfying all the requirements of illuminating-gas manu-

facture. Most of the coal used hitherto has come from western Pennsylvania, the quantity supplied by other fields being relatively small. The introduction of gas coals from new or little-known districts, because of the lack of necessary testing stations and of scientific study of the complex process of gas manufacture, has been difficult.

The Michigan Gas Association took a step in the right direction when, in continuation of its cooperative research work with the department of chemical engineering of the University of Michigan, it erected in 1906, at Ann Arbor, Mich., a complete experiment station for the study of illuminating-gas manufacture. Much patient experimenting was done in designing this plant, and many changes were made in its operation before results were procured which were intelligible and could be compared with those obtained at gas works. The experiment station, as it now stands, represents the best thought of many gas engineers who freely gave their time to perfecting it.

When the technologic branch of the United States Geological Survey took up illuminating-gas investigations, it found that the pioneer work done at this plant was of much importance. Accordingly, an arrangement was made with the University of Michigan whereby Alfred H. White, professor of chemical engineering, who had directed the work at the plant, assumed charge of the Survey's investigations. For convenience and economy the work was carried on at Ann Arbor, Mich., with the consent of the Michigan Gas Association and the Ann Arbor Gas Co.; the latter had provided much of the original installation and furnished certain facilities whereby the tests were made on a commercial basis.

The investigations were so directed as to permit the testing of coals from different parts of the United States. The results show that certain coals from which good yields of gas had been expected can not be considered as available for illuminating-gas manufacture, whereas other coals give promise and should be investigated further.

The data contained in this report are the outcome of the cooperative work at the laboratory mentioned during parts of the years 1908 and 1909. The report is published as a bulletin of the Bureau of Mines, because the act of Congress which established that bureau provided for the transfer to it of the fuel testing that was being carried on by the United States Geological Survey.

PERSONNEL.

The investigations were under the supervision of Professor White. Perry Barker, assistant engineer of the United States Geological Survey, was assigned as principal assistant. After the resignation of Mr. Barker his work was continued by Dwight F. Smith, junior engineer. In addition, John H. Wyman and William A. Dunkley

rendered efficient service upon the tests. The mine sampling was in charge of George S. Pope, who, with K. M. Way and P. M. Riefkin, all of the United States Geological Survey, collected and shipped the coals tested.

ACKNOWLEDGMENTS.

Acknowledgment is made of advice from various members of the Michigan Gas Association in regard to the working details of the experiment station, and of helpful assistance from the officers and engineers of the Detroit City Gas Co., who willingly devoted time to the operation of the plant during the course of these tests.

DESCRIPTION OF TESTS.

COALS TESTED.

All samples taken for test were loaded under the inspection of an official of the United States Geological Survey, who also examined the mine and took mine samples for analysis. This procedure enabled those in charge of the tests to obtain samples with authentic histories and to make the scope of the inquiry much broader than that of the earlier investigations at the Ann Arbor plant.

Seventeen tests of coal from 11 localities were made, and the results are incorporated in this report. Table 1 summarizes the data relative to the samples collected.

TABLE 1.—*Source of coals tested at Ann Arbor, Mich.*

Field No.	Test No.	Bed.	Locality.	Railroad.
A. A. 1.....	18, 19, 20, 32	Pittsburg.....	Scott Haven, Allegheny County, Pa.	Pittsburg & Lake Erie.
A. A. 2.....	28, 22	Raton.....	Van Houten, Colfax County, N. Mex.	Santa Fe.
A. A. 3.....	30, 21	Upper Elkhorn...	Heller, Pike County, Ky.	Chesapeake & Ohio.
A. A. 4.....	24	Yampa.....	Oak Creek, Routt County, Colo.	Denver, North-western & Pacific.
A. A. 5.....	25, 27	Kanawha, No. 2.	Page, Fayette County, W. Va.	Virginian.
A. A. 6.....	26	Sopris.....	Sopris, Las Animas County, Colo.	Colorado & Southern.
A. A. 7.....	29	No. 2 or lower.	Hanna, Carbon County, Wyo.	Union Pacific.
A. A. 8.....	33	Thompson.....	Blotson, Bibb County, Ala.	Mobile & Ohio.
A. A. 9.....	34	Saginaw.....	Saginaw, Saginaw County, Mich.	Michigan Central.
A. A. 10.....	31	Rex.....	La Follette, Campbell County, Tenn.	Louisville & Nashville.
A. A. 11.....	35	No. 5.....	Harrisburg, Saline County, Ill.	Big Four.

SUMMARY OF RESULTS.

The material here presented constitutes a progress report of results obtained in testing different coals to determine their value for use in gas works. The results are to be taken as tentative and suggestive, and not in any way as final. Indeed, since a coal that would give good results in one gas works might not prove satisfactory under the differing conditions prevailing at another, a testing station should

hardly be expected to make positive statements as to the value of a given coal. The purpose of such a station should rather be to test the coal under diverse conditions and to find what conditions give the best returns. The work here set forth falls short of that ideal chiefly because of the scantiness of the data. Gas coals should be tested in the retort at both high and low temperatures, and the tests should be repeated until it is fairly certain that no serious error has been made; but since a long time is required for a complete study of a number of coals, it has seemed wise to present the data collected on these 11 coals with only a slight interpretative discussion and to leave the reader to draw such conclusions as seem warranted.

YIELD OF COKE.

Although the results given in this report have been studied from various standpoints and have suggested some interesting deductions, the only conclusion which seems to apply to all coals well enough to warrant presentation is that relating to the percentage yield of coke. This percentage is only slightly dependent on retort temperature, for the retort is always hot enough to drive off the volatile matter in a coal, and the secondary changes of the distillation products at high retort temperatures do not greatly affect the yield of coke.

As is to be expected, the yield of coke is roughly proportionate to the ratio of the total fixed carbon to the volatile matter of the coal. From calculations based on the ratio:

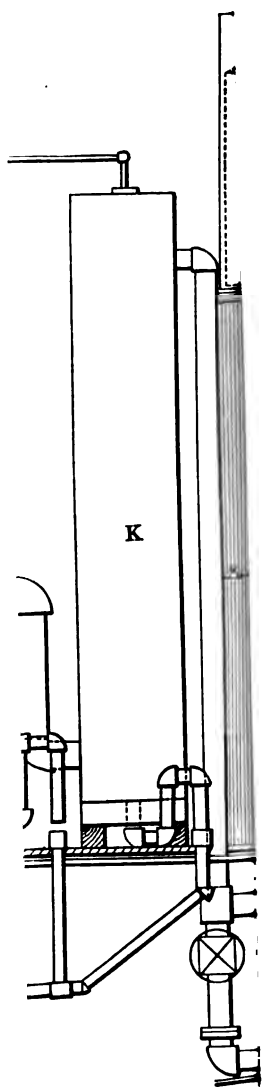
$$\frac{\text{Fixed carbon} + \text{ash}}{\text{Moisture} + \text{volatile matter}}$$

the rank shown in the table below is obtained. The agreement shown between the computed ratio and the actual percentage of coke is close, the only coal not in agreement being the one from New Mexico.^a

Coke yield of coals tested.

Rank of coal.	Fixed carbon+ash Moisture+volatile matter	Percentage of coke from coal as charged.
1. Wyoming.....	0.82	50.2
2. Michigan.....	1.43	59.5
3. Oak Creek, Colo.....	1.53	60.0
4. Illinois.....	1.55	62.3
5. Tennessee.....	1.58	66.8
6. Pennsylvania.....	1.82	67.0
7. Kentucky.....	1.88	67.0
8. New Mexico.....	1.93	68.9
9. Alabama.....	2.04	68.3
10. West Virginia.....	2.21	73.3
11. Sopris, Colo.....	2.90	74.9

^a Errors in one of the two tests, 22 and 23, made with this coal are probably responsible for the coal not being in agreement. The weight of coal charged in test 23 may have been wrongly reported. See page 36.



ment station.

VALUE OF A TESTING STATION.

It seems safe to say that the most important contributions thus far of the testing station to illuminating-gas investigations have been its own development and the proof that it can yield reliable results. The cost of equipping a similar plant at a gas works is not high, and valuable returns can be obtained without the large testing crew required by the efforts made in these experiments to collect all possible data. Seemingly, it would be well worth while for individual gas works to install such plants for working out their own peculiar problems.

EQUIPMENT OF PLANT.

GENERAL STATEMENT.

The gas was taken from a single retort of the standard D-shape, 26 inches wide by 16 inches high by 9 feet long, one of the upper retorts (A, figs. 1 and 2) of a bench of six with three-quarter depth regenerative setting. When the plant was in operation the valve C was closed and D was opened, allowing the gas to pass into the con-

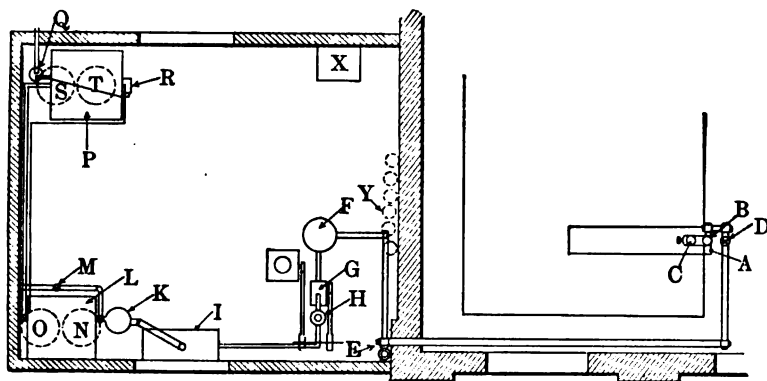


FIGURE 2.—Floor plan of illuminating-gas experiment station.

densing system E. The condensers were so equipped that either water or steam could be passed through the cooling jackets, thereby insuring a fairly accurate control of the gas temperature.

From the condensers the gas was drawn through a tar separator (Pl. I; fig. 3; also F, figs. 1 and 2) of the Pelouzé and Audouin type and constructed especially for this plant, by a blower (Pl. II, A, and G, fig. 1) driven by a 4-horsepower vertical engine. This blower was placed beyond the tar extractor in order to eliminate any error from its possible action as a tar separator. In a by-pass around the exhaustor was a pressure regulator, H, so counterpoised that a nearly uniform pressure was maintained at the retort.

The next piece of apparatus (fig. 4 and I, figs. 1 and 2) was a tar washer of the standard type. From the tar washer the gas passed into a tower scrubber (Pl. III, and K, figs. 1 and 2), which was made

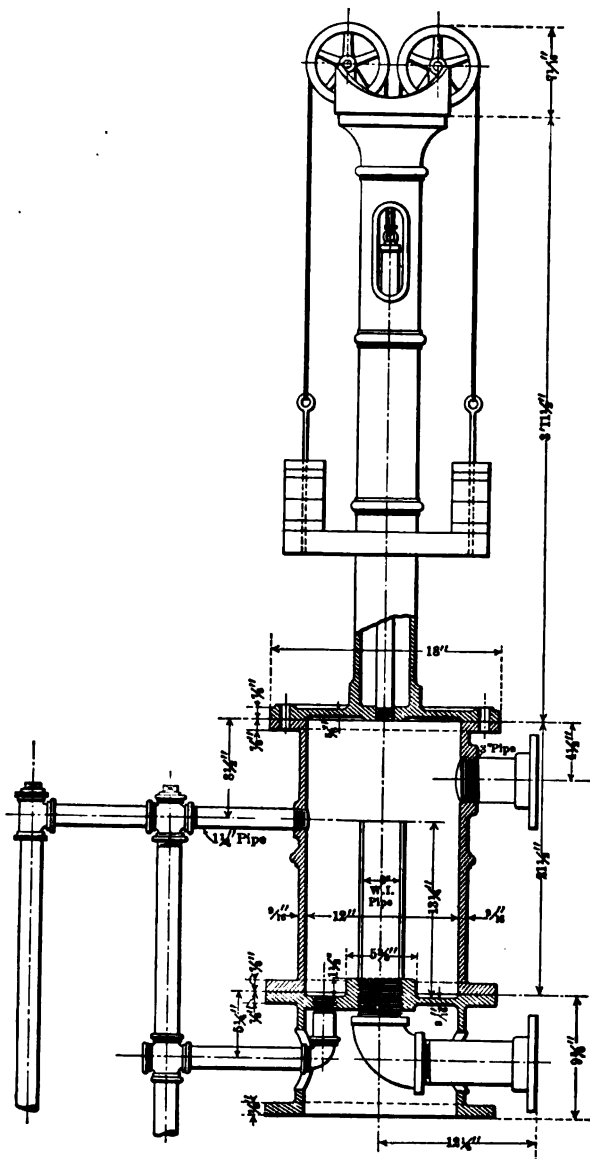
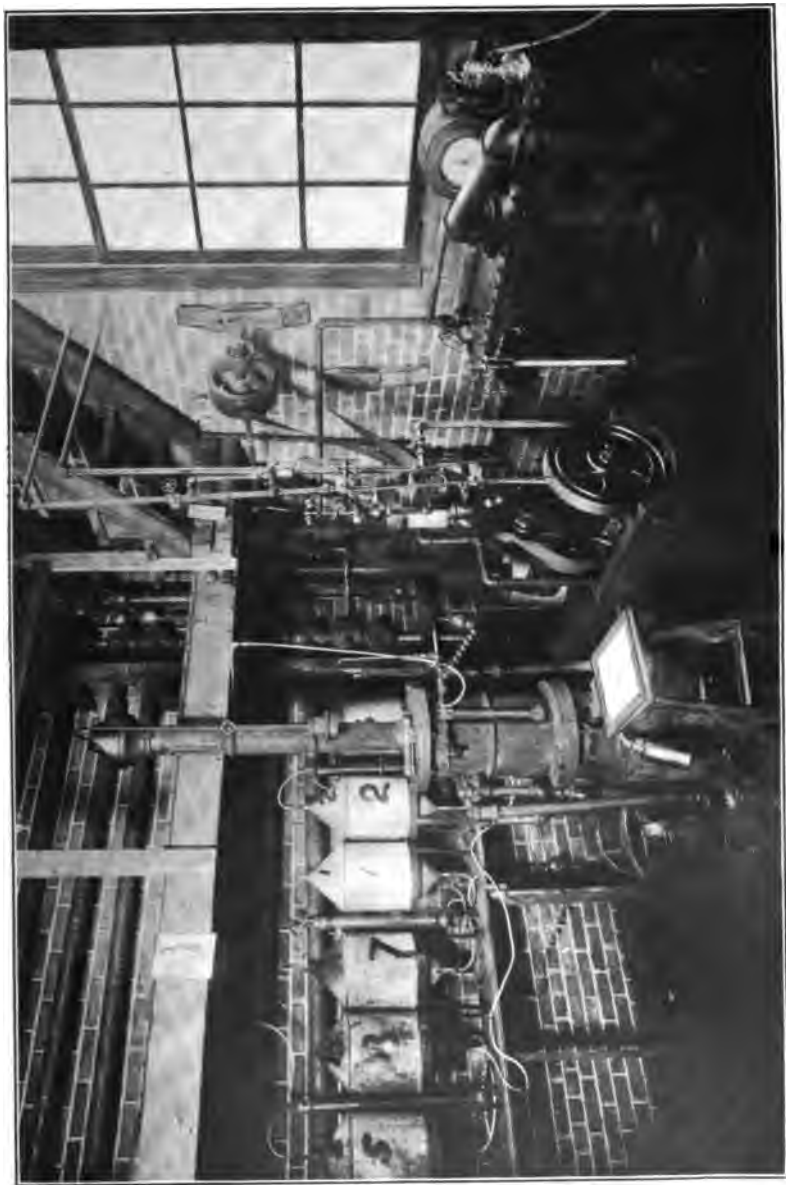


FIGURE 3.—Sectional elevation of tar separator.

from a piece of 16-inch water pipe filled with coke and fitted with a water spray. A 3 by 3 foot purifying box (fig. 5; also L, figs. 1 and 2), provided with a by-pass, M, and a 3-foot station meter, P, completed the equipment.



CONDENSERS, TAR SEPARATOR, ENGINE, EXHAUSTER, AND TAR WASHER, WITH ASPIRATOR TANKS AND APPARATUS FOR SAMPLING AMMONIA AND NAPHTHALENE IN POSITION.

The condensers (Pl. I) were in three vertical units, and had annular spaces in which either cold or hot water could be circulated so that the gas could be brought to the inlet of the tar separator under fairly well controlled conditions.

REGULATION OF PRESSURE IN RETORT.

As the quantity of gas to be handled was small and varied during the test from a possible maximum of 12 feet per minute to a minimum

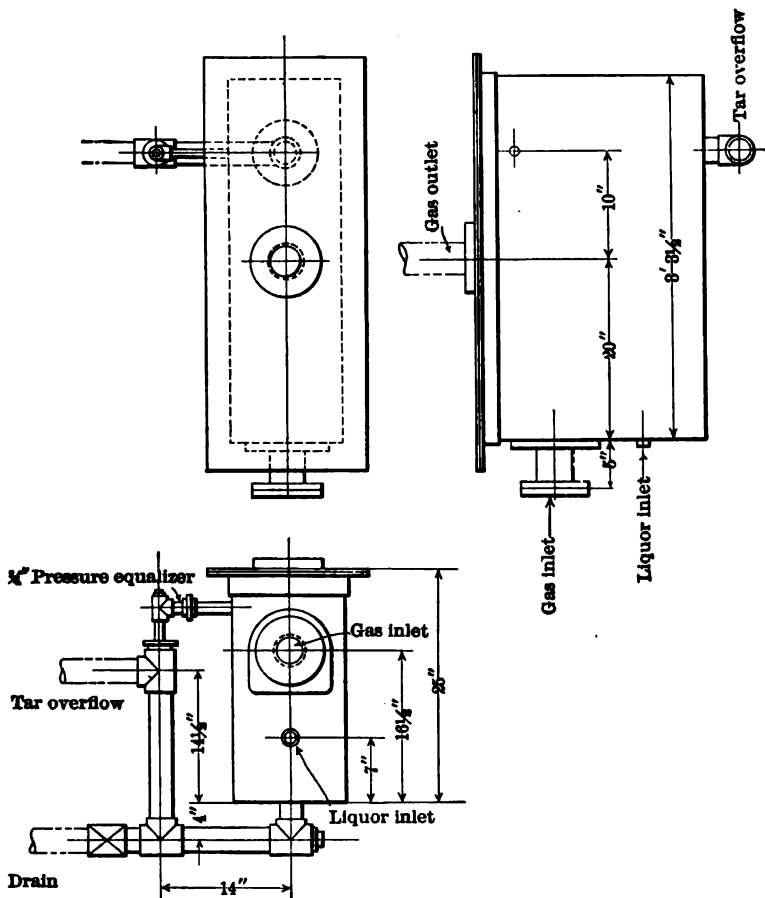


FIGURE 4.—Tar and ammonia washer.

at the end of the run of 1 foot per minute, trouble was experienced in obtaining a satisfactory exhaustor. The slow-speed exhaustor that was first used transmitted positive and negative nodes of compression which produced rapid fluctuations of pressure, equivalent to 1 inch of water, at the retort. Three different exhaustors and several modifications of governors were tried in the effort to overcome this difficulty.

In the final arrangement a small blower, run at about 400 revolutions per minute, had a primary by-pass operated by a hand wheel for rough regulation and a secondary by-pass controlled by a modified diaphragm governor (H, fig. 1), which was actuated through a $\frac{3}{4}$ -inch pipe by the pressure of the gas in the retort. This arrangement was not perfect, for the leather diaphragm soon stiffened by contact with the unpurified gas and did not respond just as quickly as the fluctuating speed of the engine required; but with care it was possible to operate the regulating device so that the retort pressure never for

over a minute at a time varied more than one-tenth of an inch from the one-tenth of an inch back-pressure intended to be kept on the retort.

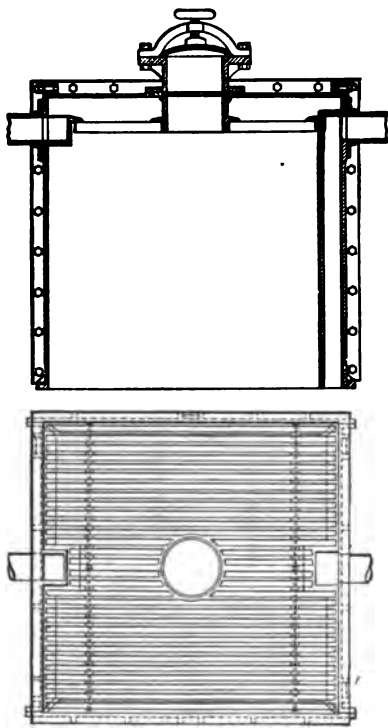


FIGURE 5.—Purifying box.

GAS-SAMPLING DEVICES.

Much attention was paid to improving the accuracy of the various gas-sampling devices. The tanks which sampled the gas by half-hour periods (1 and 2, Pl. I, and S and T, fig. 1) were made, with slight modifications, according to the design published in the proceedings of the American Gas Institute for 1907 (p. 443), and gave relatively little trouble, for there was not much variation in the rate of production nor in the quality of the gas during a single half hour. The reverse was true of the proportional tank (Pls. II, B, and III, and O, fig. 1), which was supposed to collect a mixed sample representing the gas made from beginning to end of the

run. This tank was filled by a small wet meter that worked as a pump and was geared to the shaft of the station meter, as shown in Plate III and at R in figure 1, so that an aliquot part of all the gas made was forced by the meter to the proportional tank. A diaphragm pressure regulator (U, fig. 1) kept a constant pressure on the inlet of this small meter.

So far as appearances went the pump and pressure regulator worked well, but the chemical composition and heating value of the gas collected by the proportional tank did not compare as they should with the corresponding figures for the average of the samples from the half-hour tanks. The proportional tank usually gave high



4. DETAIL OF EXHAUSTER, SHOWING BY-PASS AND DIAPHRAGM PRESSURE REGULATOR.



B. STATION METER, PROPORTIONAL METER, AND GAS TANKS 1 AND 2.

results, which is the reason for the belief that the proportional meter lagged at the extremely slow speeds at the end of the test and did not deliver the proportional quantity of the poor gas then made. It is possible, also, that incomplete mixing of the heavy, rich gases at the bottom of the proportional tank with the light, poor gases at the top caused part of the trouble.

LABORATORY APPARATUS.

The laboratory equipment included an excellent bar photometer, a Junker calorimeter, a complete gas analysis outfit, a Drehschmidt apparatus for determining the total sulphur in the purified gas, two Hoskins pyrometers and a Morse thermogage for optical measurement of retort temperatures.

Some of the chemical work was done at this laboratory and some at the university laboratory. The analyses of all the coals collected and of all the cokes produced were made at the Pittsburgh laboratory of the United States Geological Survey.

PROCEDURE OF TESTS.

Three tons of coal were shipped as a sample from each mine, usually in canvas sacks holding 100 pounds each. Some of the coal was screened before shipment, but as many of the mines had no screens much of the coal was forwarded in run-of-mine form. Rough handling during shipment broke up some of the coals considerably. To make all tests comparable in respect to size of coal charged into the retort, all coals, except on the first few tests, were screened with a standard screen such as is used at the mines in preparing gas coal. This screen, 2 feet wide and 6 feet long, was made of one-fourth inch bar iron spaced to three-fourths inch and set at an angle of 30°. The fine coal which fell through the screen was rejected, although it was frequently sampled and analyzed to determine the effect of screening on the separation of impurities.

Before starting a regular test on any given coal a preliminary charge of 400 pounds of the coal was carbonized for four and one-half hours to fill all parts of the condensing system with the distillation products and to prove that everything was in order. No test observations were taken in this preliminary run and the condensed tar and liquor were allowed to flow into the tar well of the city gas works. Meanwhile the jars for the collection of tar and liquor were tared and tagged, the gas-sampling tanks and vessels were filled with water, the sampling apparatus for determining ammonia and naphthalene in the gas was put in place, the coal for the test proper was weighed, and the pyrometer for measuring the temperature in the fire space surrounding the retort was put through the hole in the end wall of

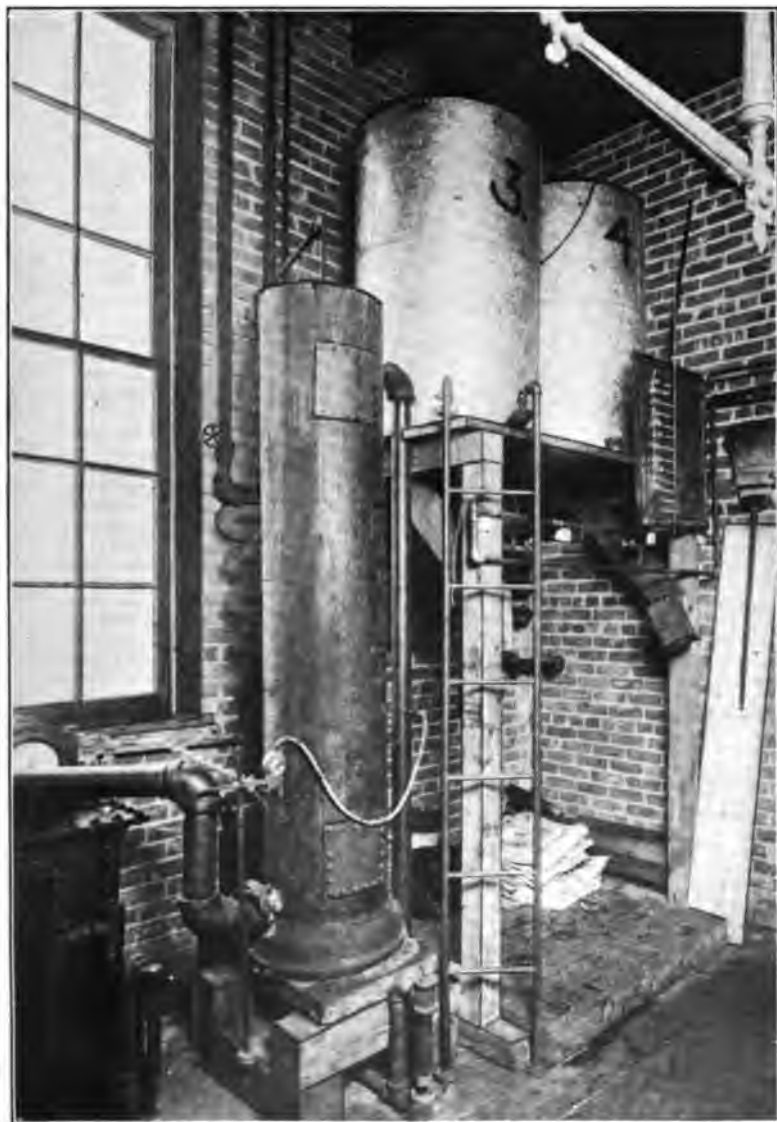
the setting (1, fig. 1) until the thermocouple was about $1\frac{1}{2}$ inches from the middle of the retort.

While the coke of the preliminary test was being drawn, the valve into the tar well was closed, the tar that had accumulated in the Pelouzé & Audouin tar separator was drawn off and the tar separator itself filled to overflowing with fresh ammoniacal liquor from the works; the jars for the collection of tar and liquor were put in place on the condenser (*a* and *c* of fig. 1), and the hands of the station meter were set at zero.

While this was being done in the experimental plant, an operator in the retort house was supervising the charging of the test retort. The retort was first examined to make sure it was clean and in serviceable condition; then the car, which contained the exact weight of coal to be used, was brought up and the coal was charged by the regular stokers in the usual manner. As soon as the mouthpiece was closed the rise in pressure shown by the gage in the experimental plant notified the operator there, who at once started the engine. The pyrometer to measure the temperature on the inside of the retort (2, fig. 1) was put through the hole in the mouthpiece and pushed back until the unprotected thermocouple rested on the coal halfway from the front to the back of the retort. The pyrometer was then luted in. After the operator had made sure that the mouthpiece and all connections were tight, he returned to his station.

In the experimental plant the speed of the engine was so regulated that the manometers indicating the pressure on the mouthpiece of the retort and at the inlet of the tar separator (*d*, *h*, fig. 1) showed a back pressure of about one-tenth of an inch of water. So long as the manometers gave concordant readings, there was free communication between them. After the pressure had been regulated by the engine throttle and the gate valve on the by-pass of the exhauster, the diaphragm regulator controlled it automatically. As the yield of gas dropped off near the end of the run, it was necessary to open the gate valve on the by-pass more and more, thereby allowing the gas to circulate more freely through the by-pass and diminishing the suction on the retort.

The collection of tar and ammonia liquor in jars (*a*, *c*, fig. 1) and the registration of the gas started automatically as soon as the retort was closed. The apparatus for testing the ammonia in the gas at the inlet of the tar separator (Pl. I and *b*, fig. 1) and the naphthalene in the gas at the outlet (fig. 1, *f*) were also started promptly at the beginning of the test, as was the H_2S sampling apparatus. However, since the system had filled with air while the coke of the preliminary charge was being drawn, a five-minute interval was allowed to elapse before the gas for candlepower and heat-value tests was turned into the sampling tank S or T.



AMMONIA SCRUBBER AND TWO OF THE GAS TANKS, WITH APPARATUS FOR DETERMINING HYDROGEN SULPHIDE IN POSITION.

The division of duties among the testing crew was as follows: One man read the station meter every five minutes, and also recorded the pressures shown by the gage (H, fig. 1) on the diaphragm governor line from the standpipe, the pressure at the inlet of the tar separator F, the differential pressures on the separator *e*, and tar washer I, and the pressure at the meter outlet *k*. A second man at five-minute intervals read temperatures at the inlets of the three units of the condensing system 4, 5, and 6, at the inlet and outlet of the tar separator 7 and 8, and took the temperature of the water of the station meter 9 and of the gas leaving it at 10. This man was responsible for the regulation of the condensers. By the use of cold water for the condensers it was possible to keep the temperature at the inlet of the tar separator from rising above 120° F. after the first few minutes of the test. As the test progressed and less steam came over, the water was cut off. Toward the close of the test steam was turned on the condensers, and, at the last, even with full steam it was not possible to keep the temperatures above 100° F.

METHODS OF SAMPLING AND ANALYSIS.

The sampling of the ammonia, tar, and naphthalene in the gas at the tar separator kept one man extremely busy and requires further description.

AMMONIA.

Sampling.—It was necessary to deviate from the practice, usually followed at gas works, of determining the quantity of ammonia in the liquor flowing from the separator and scrubber, because of the large volume of liquor in these devices at the beginning and the end of a test as compared with the volume collected during the test period of five hours. The ammonia liquor from the condensers was collected each half hour and the ammonia in the gas from the condenser was determined from samples taken at the inlet of the tar separator. The sampling was done by drawing a portion of the gas through bottles containing dilute acid. The aspirator tanks used for the purpose held about a cubic foot of gas, and were adjusted to empty in a half hour, the exact time at the start and the end of each half-hour period being recorded. After the ammonia in the half-hour sample was determined, the quantity of ammonia in the whole volume of gas was computed from the station-meter readings, allowance being made for the few minutes' interval between collection periods. These figures were then, for ready comparison, calculated to even half-hour intervals.

Analysis.—The ammonia in a measured volume of gas at the tar-separator inlet (*b*, fig. 1) was absorbed in 7 per cent sulphuric acid contained in the Wouff bottles, into which it was drawn by an

aspirator of known capacity. The acid solution was then made alkaline, and the ammonia liberated was absorbed in a known volume of standard acid. The acid not neutralized by the ammonia was found by titration against standard alkali. The ammonia in the liquor drawn from the condensing system was determined from weighed 20-gram samples by neutralization and distillation as above.

SAMPLING AND ANALYSIS OF TAR AND GAS FOR NAPHTHALENE.

The tanks, N, S, and T, were connected to the outlet of the large meter and one was filled during each half hour. The connections to these tanks were so arranged that any one tank could be connected either with the gas mains at the meter outlet or with the lines leading to the laboratory, as shown by figure 1. On account of the great analytical labor involved, the naphthalenes in the tar fog and gas at the inlet of the separator were determined only a few times, and those at the outlet in only about half the tests. For the same reason the naphthalene that had been dissolved by the tar in the condenser and the tar separator was determined only in an average sample of the combined tars for the whole distillation period.

The naphthalene in the gas at the outlet, and sometimes at the inlet, of the tar separator was determined by scrubbing or passing a known volume of gas through a saturated picric-acid solution containing 10 per cent of acetic acid, the tar fog being filtered out by an asbestos filter. This tar tube and picric-acid solution were changed as often as the aspirator emptied, and were later removed to the laboratory for analysis. A large excess of alkali was subsequently added to these solutions and the free naphthalene was expelled at 70° C., drawn into a weighed U tube, and there condensed at 0° C.

The naphthalene in the tar suspended in the gas at the inlet and the outlet of the tar separator was determined by heating samples of tar from known volumes of gas to 70° C. until no naphthalene was given off. This naphthalene was drawn into weighed U tubes and there condensed at 0° C.

COLLECTION OF SAMPLES.

For collecting the samples of tar and gas a glass tube about 6 inches long containing fibrous asbestos and glass wool, a bulb tube containing picric-acid solution, and a calibrated aspirator were placed in series and the gas drawn through the system at different rates, determined by the point of sampling. For the standpipe samples, most of which were drawn 4 or 5 inches from the top of the standpipe or on the bridge pipe, an aspirator of about one-third cubic foot capacity was used and the gas was drawn into it at a rate of about

one-third cubic foot per hour. The weighed tube containing the asbestos and glass wool was inserted less than half its length in the hole tapped in the standpipe, because of the high temperature of the gas at this point. When the tube was placed inside the pipe a great deal of tar and oil passed uncondensed through the filter to the picric-acid bulb tube. For taking samples at other points than the standpipe a cubic-foot aspirator was used, the time of drawing a cubic foot of gas being two or three hours. Tar tubes for all these samples were placed inside the main, because the samples collected there would be most comparable to those collected in mains under service conditions.

ANALYSIS.

The analysis of the tar collected in the asbestos filter was conducted as follows: The tube containing tar and water incorporated in the asbestos was weighed, this weight giving by difference the weight of tar and moisture collected. The contents of the tube and, if the tar stuck to the glass, pieces of the glass itself were put into the

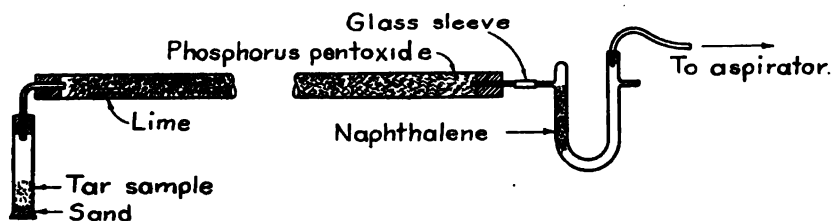


FIGURE 6.—Apparatus for analysis of tar.

volatilizing tube, which was then weighed. The tube was placed in series with the drying train and the naphthalene U tube, as shown in figure 6, and the whole system was put in the oven. The oven was heated to 70°–80° C. and air was slowly drawn through the system, volatilizing the naphthalene and moisture in the tar. The moisture was taken up by the drier—lime and phosphorus pentoxide—and the naphthalene passed on to be frozen out in the U tube, which was placed just outside the oven in a trough containing ice water. The analysis was complete when the weight of the naphthalene U tube remained constant, or very nearly so, at consecutive weighings made after a two or three hour interval. The time required for an analysis varied with the quantity of tar in the sample, but was usually 30 or 40 hours for standpipe samples containing 8 to 10 grams of tar.

When the analysis was completed, the volatilizing tube was again weighed; the loss showed the weight of the moisture and naphthalene given off. From the weight of the naphthalene in the U tube was determined the weight of the moisture also. The moisture determination, however, was at best only approximate, because more or less of the light oils, such as benzol, always escaped from the tar with the

moisture and naphthalene, and the laboratory had no means of determining them. Finally, the volatilizing tube was placed in a Soxhlet extractor and the remaining contents were extracted with chloroform until free of all soluble material. After the tube was dry it was weighed to find the weight of free carbon in the tar.

APPARATUS.

The oven used was of galvanized iron. It was 20 inches high by 16 inches wide by 14 inches deep, and was so arranged that eight samples could be worked at a time. The volatilizing tube was made from a 6-inch test tube cut in two about 4 inches from the top. A piece of muslin was wired over the flare, a layer of sand about one-half inch deep was put in, and the tube was then ready for the contents of the tar filters. The drying train (fig. 6) consisted of a heavy glass tube of about one-half inch internal diameter and 12 inches long, which contained broken lime for about two-thirds of its length and phosphorus pentoxide thoroughly incorporated in glass wool the other one-third. The glass wool prevented the gas from forming channels in the phosphorus pentoxide, and thus aided in rendering the extraction of moisture complete before the gas reached the naphthalene U tube. In order to avoid an excessive expense for phosphorus pentoxide it was necessary to use lime that had an extremely rapid reaction with water. If the lumps of lime were too small, their expansion while slaking cracked the tube. If the lumps were too large, the gas was not dried enough. Connection with the naphthalene U tube was made, as shown in figure 6, through a glass sleeve rendered air tight by an inclosing piece of gum-rubber tubing. The glass kept the naphthalene from coming in contact with the rubber.

REMARKS.

This method of analysis permitted the estimation of water, non-volatile tar, free carbon, and naphthalene. The separation of the water and the anhydrous tar was not very exact, any light oils that were vaporized by the air drawn through the train being reported as water. The determination of naphthalene, however, was precise enough. The air left the drying system saturated with naphthalene at the temperature of ice water, but this loss need not have amounted to over a milligram for each 10-hour run. It is possible that the naphthalene might have been contaminated by other hydrocarbons and, as has been pointed out in an account of previous work at the plant, the naphthalene deposit was sometimes slightly oily and had a low melting point. An attempt was made to determine the nature of this oil, but the small quantity collected rendered the attempt difficult. The oil has been shown to be an unsaturated hydrocarbon,

and it is quite possible the little-known indene, C_9H_8 , a transition product between benzol and naphthalene that has been isolated from the tar distillates which contain most naphthalene. The oil has been found rather abundantly in the analyses of some tar distillates, as would be expected of indene, but is much more plentiful in some tars than in others. In any case, however, the reporting of this oil as naphthalene seemed justifiable, because the oil would be found associated with naphthalene and would separate out with the latter in a gas plant just the same as in the laboratory. It is possible that the presence of this oil may aid in preventing solid deposits of naphthalene, and its occurrence is worthy of note.

HYDROGEN SULPHIDE IN PURIFIED GAS.

The hydrogen sulphide in the gas leaving the ammonia scrubber (j, fig. 1) was determined, in the later tests, by bubbling a portion of the gas slowly through ammoniacal cadmium chloride and then through an experimental meter. The usual lead-acetate test never disclosed the presence of hydrogen sulphide in the purified gas.

YIELD OF COKE.

The standard test charge of 400 pounds of coal was usually carbonized in four and one-half hours, but in case a longer time was needed, carbonization was continued till the evolution of gas from the charge dropped to about 1 cubic foot per minute. The coke was drawn by the stokers into a special buggy placed just below the mouthpiece of the retort, and weighed at once without quenching. It was then quenched, allowed to stand in the buggy for two or three days, and reweighed. The expectation was that some idea would be obtained of the relative quantities of water retained by the cokes from the various coals. Unfortunately, the cokes had to be drowned so completely to prevent their catching fire that they absorbed excessive quantities of water, which they still retained at the second weighing, so that the moisture figures for quenched coke have little value. The wet coke was screened over a three-fourths-inch bar screen in the same manner as the coal, and a representative sample was sent to the Pittsburgh laboratory for analysis.

The analyses reported as dry coke are taken to represent the coke as it was drawn from the retort before quenching, although the analyses of coke as received by the laboratory do not necessarily correspond to the quenched coke, because of an unavoidable loss of moisture during sampling. The percentage of coke substance is figured from the analysis and represents coke free from ash and moisture.

YIELD AND QUALITY OF GAS.

COLLECTION OF SAMPLES.

The two tanks for sampling the purified gas were used alternately and were changed exactly at the close of each half hour. Error due to solubility of the varying gases in the water of the calorimeter and photometer meters was avoided, so far as was possible, by allowing the gas to flow through the meters for 15 minutes to saturate the water before starting the test.

CANDLEPOWER.

The candlepower of each half-hour sample was determined by means of a Lummer-Brodhum contrast photometer mounted on a 2,500-mm. bench. This photometer was equipped with a Sugg D burner and Hefner amyl-acetate lamp. The Sugg D burner was chosen from among the many photometric burners because it is frequently used and does not require the complicated adjustment of the new Metropolitan No. 2 burner, which would probably have been adopted but for the need of rapid work and the widely varying candlepower of the gas. The burner was regulated to burn about 5 feet per hour, although its rate was reduced to 4 feet for the high candlepower samples from the first part of the distillation period.

ANALYSIS.

A sample of gas representing each half-hour period was collected in a special gas holder of the Hempel type, having both inlet and outlet sealed with water columns of small cross section and closed with rubber tubing with screw clamps. If the water previously in the holder had been saturated with gas, and the holder had been so filled with gas that the water and gas were in contact only in the narrow tubes at the top and bottom of the holder, a sample could be kept in a holder for several days in perfect safety.

Gas analyses were made on the half-hour samples by means of the regular Hempel pipettes. Water saturated with illuminating gas was employed as a containing liquid in the measuring burettes. Bromine water was used as an absorbent for illuminants, oxygen was removed by phosphorus, and methane and hydrogen were determined by the explosion method.

The lead-acetate test did not show the presence of hydrogen sulphide in the purified gas.

Only the sulphur present as hydrogen sulphide in the unpurified gas was determined. In the method used the unpurified gas was passed at the rate of about 0.2 foot per hour through an ammoniacal solution of cadmium chloride contained in a Meyer absorption bulb.

MEASUREMENT OF RETORT TEMPERATURES AND PRESSURES.

In tests 18 to 30, temperatures within the retort were taken with a Bristol pyrometer having a 9-foot fire end. The junction of the couple was placed on the surface of the charge $4\frac{1}{2}$ feet from the mouth-piece. In tests 30 to 35, a Hoskins heat gage was used in a similar manner. In tests 18 to 30, the temperatures of the fire space outside the retort were taken with a platinum-rhodium couple inserted into the fire space at 1 (fig. 1), with its junction 2 inches from the retort. In tests 30 to 35, a Hoskins heat gage was used for this purpose. The temperature of the inner surface of the retort walls was sometimes taken before charging and after drawing by means of a Morse thermogage. All pyrometer readings were corrected by frequent calibration at the university laboratory against a standard platinum, platinum-rhodium couple. Temperatures at points 3, 4, 5, 6, 7, 8, 9, and 10 (fig. 1) were taken with thermometers (Fahrenheit scale) calibrated in a water bath against a certified German standard thermometer.

Pressures at *d*, *e*, *h*, *i*, and *k* are expressed in tenths of an inch of water. Notations are made in such cases, as are differential pressure readings.

WEIGHT BALANCE.

The weight balance of the distillation process is the sum of the products recovered plus the loss unaccounted for. The weights of coke, gas, tar, and ammoniacal liquor collected are added, and to this sum is added the computed weight of water vapor, which should condense in the gas separator when the gas is cooled to the standard temperature of 60° F. The computation is readily made by using tables of vapor tension. The difference between the sum of the known products and the weight of the original charge is reported as "Loss unaccounted for."

BASIS OF COMPUTATIONS.

Because this report presents a comparison of the products of the destructive distillation of coals of different composition, especial care has been taken to set forth the characteristics of the products as fully as possible. Many persons consider that such a study should be based on the unit weight of coal charged. Others prefer to base computations on the unit weight of dry coal, but if the study is to be made in a scientific way, the best basis is the unit weight of coal free from moisture and ash.

COINCIDENCE OF OBSERVATIONS.

All operations during a test were performed and all readings were taken, so far as was possible, at the same time at half-hour intervals. It was feasible to collect by exact intervals the samples of tar and ammonia liquor, and also the tank samples of purified gas for candle-

power and heating value determinations and for analysis. The emptying of the tanks used for collecting samples for ammonia and the naphthalene determinations sometimes took more and sometimes less than 30 minutes, but as the observer recorded the exact times at which the emptying began and was complete, it was possible, by reference to the volume of gas registered by the station meter in the interval, to recalculate the results to even half-hour periods without any error other than that involved in the assumption that the constituents being tested had been evolved at a constant rate throughout a given half hour. This assumption might have caused a slight error in the half-hour figures, but could not have affected the totals.

PRESENTATION OF RESULTS.

The important data showing the progress of distillation by half-hour intervals for each coal have been platted in Plate IV and have been brought together for comparative study in Tables 3 to 7. In Plate IV there is given under each test the temperatures inside and outside of the retort, the candlepower and the heating value of the gas, and the variations in the percentages of the important constituents—"illuminants," hydrogen, and methane. The percentages of carbon dioxide, carbon monoxide, oxygen, and nitrogen in the gas were not tabulated nor platted, as they changed little during the distillation.

The total yields of tar and ammonia liquor are shown on the chart by half-hour intervals, as are also the yield of ammonia (NH_3) and the weight of naphthalene vaporized in the gas and dissolved in the tar at the outlet of the tar separator. This weight of naphthalene was chosen for presentation because the naphthalene that gets past the tar separator, either vaporized or dissolved, causes trouble, as dissolved naphthalene is set free by the ammonia liquor in the scrubbers.

VERIFICATION OF DATA.

GENERAL STATEMENT.

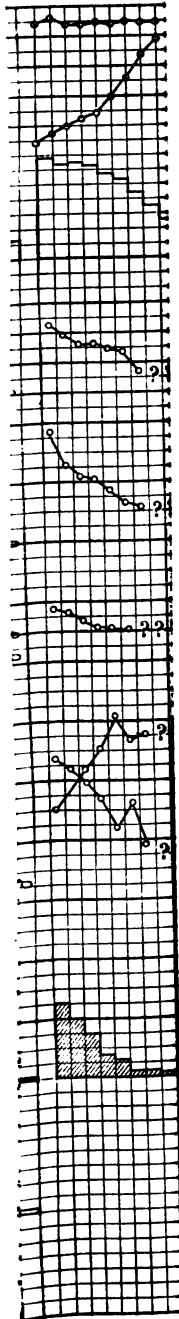
The preceding presentation of the methods of making tests and computing results should be supplemented by a discussion of the means of determining the accuracy of the results obtained.

There were several ways in which the accuracy of the work might have been checked. One method used was to take a standard coal that should give known results. In order to check the operation of the experiment station plant in this manner, a standard Pittsburg gas coal was procured and the first three tests, as well as a later one, were made on it. The results are discussed in their appropriate places.

SCOTT HAVEN, R

A.A.I. TEST 18,

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Another means used for checking results was by comparing duplicate tests of the same coal. An exact duplication of results could hardly be expected because of the number of variable factors involved, but it was sometimes possible to partly explain discrepancies as caused by differing conditions that had been noted during the test.

Valuable evidence of accuracy was obtained by studying the data gathered from a single test in the light of the knowledge gained from other tests. The platted curves showing the rate of formation of gas, ammonia, and tar should show a relation to each other and to the retort temperature, and the curves indicating the candlepower, heat value, and chemical composition of the gas should show some concordance.

A conclusive proof of inaccuracy was sometimes given by the unaccounted-for loss. The total weight of the products should always have been less than the weight of the coal charged, for there was always a loss of products while the retort was open for charging. In the results given in this bulletin some losses that swelled the unaccounted-for totals were the tar that stuck in the standpipe, of which no account was taken; the water removed from the gas in the tar separator or carried in suspension past the separator (which could not be estimated in the small plant used), and the weight of hydrogen sulphide and carbon dioxide removed by the ammonia liquor in the scrubber and purifier. The average total of such unaccounted-for losses in the tests was 6.6 per cent of the coal as charged.

Each of the complete tests recorded here required over 1,800 observations, and, as was to be expected, there may be found in the platted results of almost every test, points so far from their expected positions that they are evidently in error. Fortunately, however, determinations were made every half hour or oftener, so that an error in any one determination had a relatively small effect upon the average result.

It is believed that the care taken in the calibration of pyrometers, thermometers, gas-sampling tanks, and meters prevented any serious systematic errors in measuring temperatures, or in determining the quantity, heating value, chemical composition, or candlepower of the gas. The measuring of the quantity of tar and ammoniacal liquor from any given test is believed to have been fairly accurate, although some of the irregularities in the quantity of tar from one half-hour period to the next may have indicated a temporary stoppage in the drips, and there is a possibility suggested by the results from coals yielding stiff tars that heavy tar collected in the pipes during the latter part of the preliminary run and was washed out by lighter tar during the first part of the regular test, thus unduly increasing the tar in these samples.

AMMONIA.

There was some uncertainty concerning the yield of ammonia, because it had to be determined, as already explained, by sampling the gas at the inlet of the tar separator. The results from the separate tests should certainly be comparable with one another, and the results on the standard Pittsburg coal, though better than those usual in practice, were not high enough to throw serious doubt on their reliability, since the care taken to have the retort in good condition and to keep the exhauster working properly warranted higher than average returns of ammonia and of all other products of distillation.

The theoretical yield of ammonia was calculated from the ultimate analysis of the coal on the assumption that all of the nitrogen found might theoretically be converted into NH_3 . Fourteen parts by weight of N form 17 of NH_3 , and on this basis the Alabama coal, No. 8, with 1.24 per cent nitrogen in the dry coal, should yield theoretically 1.51 per cent ammonia, or 30.1 pounds NH_3 per ton. The actual yield of ammonia was only 4.15 pounds NH_3 per ton, or 13.8 per cent of the theoretical.

NAPHTHALENE.

No great accuracy is claimed for the naphthalene figures on account of the large correction factors involved in the tedious process of analysis. The results should be comparable with one another, but an absolute error of 10 per cent is entirely probable.

QUANTITY AND QUALITY OF GAS.

The weight of gas was computed by multiplying the corrected volume of gas by its weight per cubic foot. The latter figure was calculated from the percentage composition of the gas and the known weights of the constituents—carbon dioxide, hydrogen, etc. For this purpose the "illuminants" were assumed to be ethylene, an assumption not strictly accurate but involving only a slight error. In published tables the weights usually given of the various gases are those calculated for a temperature of 0°C . These values were recalculated to the standard conditions—gas saturated with moisture at a temperature of 60°F . and a barometric pressure of 30 inches.

The values obtained are as follows:

TABLE 2.—*Weight per cubic foot of various gases saturated with moisture at 60°F . and 30 inches barometric pressure.*

	Pounds.
Carbon dioxide	0.1140
Carbon monoxide0726
Ethylene0726
Hydrogen0052
Methane0415
Nitrogen0728
Oxygen0830

The average heat value, candlepower, and chemical composition of the gas, as calculated from the figures for the nine half hours of the test, were computed by reference to the volume of gas made during each period. Thus, if in one half hour the yield of gas was 250 cubic feet and its heating value was 650 British thermal units, the product of 650 by 250 was taken as the number of heat units produced during the period and was added to the figures similarly obtained for the other periods. The total number of heat units thus obtained divided by the total output of gas gave the average heating value.

This method gave correct averages of heating value and chemical composition. The average candlepower obtained in this way, however, was only approximate, since the candlepower of a gas is a function of two entirely distinct variables—temperature and the amount of free carbon liberated in the flame—and hence the candlepower of a mixture of gases will not usually be the arithmetic mean of the candlepower of the component gases.

COAL AND COKE.

It would seem that the data on coal and coke should be the least liable to error, and yet, partly because of the lack of sensitiveness in the scales used, partly because of the loss in charging and drawing or the possible addition of carbon knocked off the roof of the retort, and possibly because of carelessness in weighing (the simplicity of the operation causing too much reliance to be placed on a single reading), the internal evidence indicates that the largest single errors were those in the weights of the coal and coke. The most exasperating illustration of this fact is found in test 25 on West Virginia coal, where the sum of the products is 101.7 per cent of the weight of the coal. An error so large could hardly have come from any other source than incorrect weighing.

Another error in some of the tests is revealed by comparing the percentages of ash in the coal and coke. If all the experimental tests had been correctly made the calculated per cent of ash in the coke would be the same as the percentage obtained, which was not the case in all the tests. The coke sample taken was carefully crushed and quartered for analysis, but a similar procedure was not followed in sampling the coal because it was not desirable to change the proportion of coarse and fine coal in the sample as it left the screen. The weight to be sampled for each charge was so small and the size of the material so uneven that the coal sampling could not be accurate. The greatest discrepancy in the analyses was, of course, in the ash. In the same way the few sacks of coal used for one test may have had a composition materially different from that of the coal used in a duplicate test and from that of the mine sample.

Attention is called to marked discrepancies of this sort in the notes on the individual tests.

The probable reliability of each test is discussed in the description of the test.

DETAILS OF TESTS.

Eighteen tests on 11 different coals from 10 States are here reported. For convenience of reference the tabulated results are arranged in alphabetical order by States, except that the tests on the standard Pittsburg coal are discussed first because of their value as an index of the reliability of the whole series. The data concerning the analyses and chemical composition of all the coals and cokes are given in Tables 3 and 4. The condensed data of the products of distillation for all the tests are given in Tables 5, 6, and 7. The course of each test as indicated by the retort temperature, rate of gas production, etc., at half-hour intervals, is shown graphically in Plate IV.

The individual tests are discussed in the order shown by the following table:

Order in which tests are discussed.

State.	Place.	Coal bed.	Field No.	Test No.
Pennsylvania.....	Scott Haven.....	Pittsburg.....	A. A. 1	18, 19, 20, 32
Alabama.....	Blotcon.....	Thompson-Underwood.....	A. A. 8	33
Colorado.....	Oak Creek.....	Yampa.....	A. A. 4	24
Do.....	Sopris.....	Sopris.....	A. A. 6	26
Illinois.....	Harrisburg.....	No. 5.....	A. A. 12	35
Kentucky.....	Hellier.....	Upper Elkhorn.....	A. A. 3	21, 30
Michigan.....	Saginaw.....	Saginaw.....	A. A. 9	34
New Mexico.....	Van Houten.....	Raton.....	A. A. 2	22, 28
Tennessee.....	La Follette.....	Rex.....	A. A. 10	31
West Virginia.....	Page.....	Kanawha No. 2.....	A. A. 5	25, 27
Wyoming.....	Hanna.....	Lower.....	A. A. 7	23, 29

COAL A. A. 1, FROM PITTSBURG BED, SCOTT HAVEN, PA.

The mine selected to furnish the Pittsburg coal used as a standard in the tests was the Ocean No. 2 mine, working the Pittsburg bed at Scott Haven, Allegheny County, Pa., adjacent to the Pittsburg & Lake Erie Railroad. Mine samples were collected and a car was loaded with screened coal ($\frac{3}{4}$ -inch bar screen) under the supervision of G. S. Pope of the United States Geological Survey. The car was sampled on its arrival in Ann Arbor and about 2 tons were stored in a bin for the tests. The analyses of the coal and the results of the four tests made are given in Tables 3 to 7 (pp. 41 to 46) and in Table 8 (pp. 47 to 50).

The first three tests (18, 19, and 20) were made on this coal as practice tests, and later another test (32) was made to confirm the earlier work. None of these tests was complete, and blanks in the tables indicate that some of the samples were lost or that there was an obvious error in a test. No results were rejected merely because they did not harmonize with the others.

The coal used in the first three of these tests was not screened at the experiment station, but upon reaching a decision to screen all coals the coal in the last test (32) was screened. Tables 1 and 2 show that although 17.4 per cent of the coal was rejected in screening, the composition of the screened coal did not differ much from that of the unscreened. An average of all the results obtained is given in the following table:

Average results of tests of Pittsburg coal.

Coke, per cent of coal charged.....	67.07
Gas per pound of coal charged (cubic feet)	5.04
Candlepower, approximate average.....	15.50
Candle-feet per pound of coal.....	79.50
Heating value per cubic foot (B. t. u.).....	641.00
Heating value per pound of coal (B. t. u.).....	3,280.00
Gas analysis (per cent):	
Carbon dioxide.....	1.30
Illuminants.....	3.70
Oxygen.....	.80
Carbon monoxide.....	6.50
Methane.....	34.40
Hydrogen.....	48.20
Nitrogen.....	5.00
Ammonia (NH ₃) per ton of coal charged (pounds)	5.43
Tar per ton of coal charged (pounds)	155.80

The average results obtained in regular practice at the works of the Ann Arbor Gas Co. with a coal from the same general district were: Coke, 67.8 per cent (of which 8.6 per cent was breeze); gas, 4.95 cubic feet per pound; candlepower of gas, 16.9; heating value of gas, 617 British thermal units per cubic foot.

The tests can be compared best with the aid of the curves platted in Plate IV. By starting at the top of the sheet and reading down, it will be seen that the external retort temperatures varied little during a test, but that the interior retort temperatures varied a great deal. The low starting temperature of the first test (18) is known to have been caused by the pyrometer being bedded in the coal instead of lying on its surface. The curves show plainly the effect of retort temperatures on gas production. The high retort temperatures of tests 18 and 20 caused a rapid but quickly decreasing evolution of gas. The lower retort temperatures drove the gas off more slowly and evenly, but of course the carbonization period had to be longer. During test 19 the retort was badly coated with carbon, which was removed before test 20 was run. The sluggish evolution of gas in test 19 may be partly due to this coating. The coke drawn was good in all four trials, and the two screening tests made showed only 5 per cent and 7 per cent breeze.

The candlepower of the gas was fairly uniform, the gas tenaciously maintaining its candlepower during the middle portion of the run in spite of falling heat values, a characteristic of this Pittsburg coal which may partly account for the fame it has won as a gas coal. In Plate IV the curves of candlepower in the three tests averaged lie near each other, but there is an apparent discrepancy in test 20, where the highest candlepower (16) is found with the highest retort temperatures, and accompanies the highest yield of gas (5.27 cubic feet) with the lowest heating value (611 B. t. u.). The high yield and low heating value of the gas would be expected from the high retort temperature, but the high candlepower seems to be an error, since it is not in accord with the heating value or with the chemical composition of the gas. The curves showing the heat value and the chemical composition of the gas call for little comment. The decrease of the unsaturated hydrocarbons reported as illuminants, the decrease in the methane, and the increase of the hydrogen as the test progresses characterize the gas from all the coals tested.

The tar figures for tests 18 and 19 agree fairly well. The results for test 20 were unfortunately not taken, the tars being thrown out before weighing. The figures for test 19 were spoiled by the outlet of the tar separator clogging. The weight of tar collected from the condenser was normal (66 pounds per ton of coal as compared with 68 pounds, the average of the other two tests), but the tar drawn from the separator weighed only 35 pounds as compared with 111 pounds, the average of the other two tests.

The course of ammonia formation, as shown by the curves, was fairly regular. The curve for test 32 is incomplete because a stoppage of the aspirator tank outlet in the latter part of the test made the emptying of the tank so slow that determinations calculated to half-hour intervals would be misleading, although the figures for total ammonia were unaffected. In test 20 the effect of high retort temperature in cutting down the yield of ammonia at the end of the run is noticeable.

The naphthalene data are complete for test 32 only, and show a total yield of 8.1 pounds of naphthalene per ton of coal, a yield higher than that from any other coal except the West Virginia (11.4 pounds) and the Alabama (10.7 pounds). As with all the coals, a large percentage of the naphthalene was dissolved in the tar. Close attention was paid to the small quantity of naphthalene which escaped solution in the tar, since it is this undissolved naphthalene which may cause trouble. In the test with this coal the quantity of naphthalene left as vapor would have sufficed to saturate the gas at an average temperature of 63° F.

The weight balance for these tests is given in Table 8 (p. 48) and is reproduced here reduced to a percentage basis.

Heat balance of tests of coal A. A. 1.

[Percentages on weight of coal charged.]

	Test 18.	Test 19.	Test 20.	Test 32.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Coke.....	67.1	67.5	64.7	68.5
Gas.....	14.7	16.3	15.4	14.7
Tar.....	8.1	5.2	9.9
Ammonia liquor.....	3.4	3.5	3.8
Unaccounted for.....	7.6	3.1
Total.....	100.0	100.0

* Low, as previously stated.

If it had not been for the stopping of the outlet of the tar separator in test 20, which made the tar low and the unaccounted-for loss high on that test, the figures would be in fair agreement, with the unaccounted-for loss between 3 and 4 per cent.

COAL A. A. 8, FROM BLOCTON, ALA.

The sample consisted of 60 sacks of coal from the Blocton No. 7 mine, working the Thompson-Underwood bed at Blocton, Bibb County, Ala. The coal was shipped under the direction of P. M. Riefkin. Tables 3 to 8 summarize the particulars regarding this coal and the one test (33) made on it.

The coal as received was dusty, but showed no pyrite nor slate. The high percentage of coal passing through the $\frac{3}{4}$ -inch bar screen (23.4 per cent) may have resulted from frequent handling of the sacks in transit. Screening improved the quality of the coal, the ash content of the refuse being 14.50 per cent, though that of the screened coal was only 4.29 per cent. The percentages of moisture, ash, and sulphur were low, and the volatile matter was less than in most of the coals tested, being only 29.1 per cent in the coal as charged.

The regular charge, 400 pounds, of screened coal was put into the retort, which had an external temperature about the average of the tests. The internal temperature remained low, however, and gas came off slowly, the rate not reaching a maximum till the fifth half hour. At the end of the ninth half hour distillation was practically complete. The coke was reported as not having a good appearance and showing some slate; but the percentage of breeze was only 12.5 per cent, and the total yield and the ash content were about the same as for the Pittsburg coal. The yield of gas (5.18 cubic feet per pound) was good; the initial candlepower (21.8) and heating value (612 B. t. u.) were satisfactory, but the quality fell off so rapidly that the average candlepower (14) and heating value (598 B. t. u.) were rather low. These averages give by calculation 72.5 candle-feet per pound of coal and 3,100 British thermal units in the gas per pound of coal. Both values are distinctly lower than those from the Pittsburg coal.

The runs of tar and ammonia liquor were regular. The total yield of ammonia (NH_3) was low, only 4.03 pounds per ton of coal, scarcely any being evolved during the last three half hours. The poor return was due in part to the low nitrogen content (1.24 per cent) of the dry coal. The percentage recovered was also low, being 13.8 per cent of the theoretical as compared with an average recovery for all the coals tested of 16.8 per cent. The total production of naphthalene was high, but the tar was evidently able to absorb it, so that the saturation temperature of the gas leaving the tar separator was only 57° F. The weight balance with its unaccounted-for loss of 1.7 per cent is too low, but the test is believed to be reliable. The coal deserves further investigation.

COAL A. A. 4, FROM OAK CREEK, COLO.

The sample consisted of 59 bags of run-of-mine coal from the Yampa bed of the Oak Hill mine, on the Denver, Northwestern & Pacific Railroad, at Oak Creek, Routt County, Colo. K. M. Way, of the United States Geological Survey, collected the mine samples and supervised the loading of the coal tested. The composition of the coal and the results of test 24 are given in Tables 3 to 8.

The coal had a dull luster, was clean, and showed very little slate and no pyrite. Rough handling in transit may partly account for 39.5 per cent of the coal passing the $\frac{3}{4}$ -inch bar screen. Evidently, however, the ash content was not uniform, for although the two mine samples showed only 8.3 per cent and 9.2 per cent ash in the dry coal, the sample from the sacks contained 15.67 per cent ash and the correctness of the determination is checked by the 25.65 per cent ash in the dry coke. A test for coke only (24 A) gave a coke with only 12.77 per cent ash, or as low as was to be expected from the mine sample.

The coal is notable for the high percentage of moisture it holds tenaciously. Even after air drying it held 4.59 per cent, and as charged it contained 7.17 per cent. The volatile matter in the screened coal, air dried, was 34.8 per cent; though high, it does not adequately show the difference between this and the Pittsburg coal. The ultimate analysis, with an oxygen to hydrogen ratio of 2.67 for this coal against a ratio of 1.28 for the Pittsburg coal, shows the difference better.

The 400-pound sample was placed in a hot retort, but vaporization of the moisture in the coal kept down the retort temperature during the early part of the distillation period. The yield of gas was low at first, but increased as the temperature rose, reaching its maximum in the sixth half hour and then falling off rapidly. The yield was fair, being 4.81 cubic feet per pound of coal as charged and 5.2 cubic feet per pound of dry coal. The gas of the first half hour period was accidentally contaminated with air in the sampling tank, so

that the platted results of the candlepower and heat-value tests for that period are not reliable. The candlepower for the second half hour was 21.7, and the gross heat value 802 British thermal units. The calculated average candlepower and heating value, figures for the lost sample being interpolated, are 14 candlepower and 626 British thermal units. A probable explanation of the low candlepower and heating value of the gas is that the coal, calculated to an ash and moisture-free basis, contains 13.7 per cent oxygen, and Pittsburg coal contains only 7.18 per cent. This high oxygen content must cut down the percentage of hydrocarbons in the gas and increase the proportion of the less valuable oxygen compounds.

The yield of coke, calculated to a dry coal basis, was rather low, a result of the high percentage of volatile matter and the proportion of oxygen in the coal. The excessive proportion of breeze probably resulted from the great shrinkage in coking caused by the above factors.

The yield of ammoniacal liquor during the first three half-hour periods was very heavy, but after that fell off rapidly. The total yield of ammonia (NH_3) was large, 7.6 pounds per ton of coal as charged. In explanation it should be stated that the coal contained 1.59 per cent of nitrogen, calculated to a dry-coal basis, or 25 per cent more than the sample of Pittsburg coal tested. The nitrogen content partly accounts for the high yield of ammonia, but the proportion of the total nitrogen converted into ammonia (22.6 per cent) was higher than in any other test. Low temperature within the retort from the vaporization of the moisture in the coal probably caused this high recovery, as the yield of ammonia scarcely began to decline until after the seventh half hour, when the temperature in the retort was slightly above $1,600^\circ \text{F}$.

The weight balance shows an average unaccounted-for loss of 5 per cent, and the test appears reliable. Apparently this coal can not be classified as a good gas coal.

COAL A. A. 6, FROM SOPRIS, COLO.

The sample consisted of 58 sacks of run-of-mine coal from the Sopris bed, at Sopris, on the Colorado & Southern Railroad, Las Animas County, Colo. The sacks were filled from cars in the railroad yards. K. M. Way inspected the mine and shipped the samples. The composition of the coal and the data of the one test (26) made on it are given in Tables 3 to 8.

The coal as received looked clean, and contained little slate and no pyrite. It had stood shipment well and the proportion that passed through the $\frac{3}{4}$ -inch bar screen was only 13.8 per cent. The analysis of the coal as charged shows 18.1 per cent ash, but the sample probably was not representative since the coke contained only 20.66

per cent ash and the mine samples only 10.9 and 12.4 per cent ash, respectively. Even if allowance is made for the high ash, the content of volatile matter is extremely low, being only 29.8 per cent for coal ash and moisture free, whereas the Pittsburg coal shows over 36 per cent on the same basis.

The charge, 410 pounds of screened coal was put in an extremely hot retort, the hottest used in the tests. Gas production soon reached a maximum, and fell rapidly after the sixth half hour, so that the yield in the last two half hours was extremely small. The yield of coke was large, (75 per cent of the weight of the coal as charged), a natural result of the small percentage of volatile matter in the coal.

The coke was good; only 10 per cent being breeze. The yield of gas was 4.9 cubic feet per pound of coal, but the gas averaged only 12 candlepower and 614 British thermal units per cubic foot; hence the calculated candle-feet were only 58.8 per pound of coal, though the heating value was better, 3,008 British thermal units per pound of coal.

The ammonia liquor was about normal in quantity. The yield of ammonia was low, only 4.35 pounds per ton of coal charged, since the coal is poor in nitrogen. The run of tar declined rapidly after the third half hour and the total quantity was small. No naphthalene tests were made. The weight balance shows an unaccounted-for loss of 1.2 per cent.

This test is apparently reliable. A large yield of gas can hardly be expected from a coal so low in volatile matter as this one; on the other hand, the average candlepower and heat value of the gas would probably have been higher had the retort been cooler.

COAL A. A. 12, FROM HARRISBURG, ILL.

This coal was collected by P. M. Riefkin, from the No. 4 mine, at Harrisburg, Saline County, Ill., on the Big Four Railroad. The shipment consisted of 60 sacks of coal screened over a $\frac{1}{4}$ -inch bar screen. Analyses of the coal and the results of test 35 are given in Tables 3 to 8. Detailed statements of observations made and data computed are given on pages 51 to 75.

The coal as received looked dusty, but no slate or pyrites were noted. It tended to break in finger form, and on rescreening ($\frac{1}{4}$ -inch bar screen) 27.9 per cent of it passed through. The proportion of volatile matter was greater than in any other coal tested—34.4 per cent in the coal as charged, and 39.1 per cent in coal dry and free from ash. The sulphur was also high, 1.96 per cent in the coal as charged. The coal contained more oxygen and slightly more nitrogen than the Pittsburg coal.

The 400 pounds of screened coal were put into a retort, having the lowest temperature of any used in the tests and rather thickly

coated with carbon on the crown. Because of these conditions, the evolution of gas did not reach a maximum until the fifth half hour; it was practically complete at the end of the ninth half hour. The coke was dark and easily broken but gave only 6.7 per cent breeze. The yield of gas was low—4.3 cubic feet per pound of coal. The average candlepower was 15.2, and the candle feet were 65.3. It is possible that had the retort temperature been high the yield of gas would have been greater, but it is also probable that the candlepower of the gas would have been less. The heating value of the gas, 632 British thermal units per cubic foot, was fair; the value per pound of coal, 2,718 British thermal units, was low.

The quantity of ammoniacal liquor was large, partly because the percentage of moisture and combined oxygen in the coal were both somewhat above the average. It is probable also that the low retort temperature prevented the free formation of carbon monoxide and hydrogen. The ammonia yield was large, the percentage of nitrogen in the coal being somewhat above the average; the recovery was 19 per cent of the theoretical as against an average recovery of 17 per cent for all tests. The naphthalene yield was extremely small, perhaps because of the low retort temperature. The content of hydrogen sulphide in the gas at the outlet of the scrubber was 50 per cent more than in the gas from the Pittsburg coal, but was not so high as from some other coals containing much less sulphur. The weight balance shows an unaccounted-for loss of 8.8 per cent, slightly the largest of the series.

The test is apparently reliable. The coal should be tested again at a higher temperature.

COAL A. A. 3, FROM HELLIER, KY.

The sample consisted of about 2 tons taken from a carload of run-of-mine coal shipped from the Upper Elkhorn bed at Hellier, Pike County, Ky., on the Chesapeake & Ohio Railroad. The mine samples were collected by G. S. Pope, who also saw to the shipment of the coal. The composition of the coal and the results of tests 21 and 30 are given in Tables 3 to 8.

The coal as received contained some slate and a large proportion of small-sized material. With a $\frac{1}{2}$ -inch bar screen the screenings came to 30.8 per cent, and with the regular $\frac{3}{4}$ -inch bar screen to 48.5 per cent. The analysis does not differ much from that of the standard Pittsburg coal, although the percentage of oxygen is higher.

In test 21 a light charge, 360 pounds, was placed in a rather hot retort on the advice of the gas-works foreman who had tried some coal from the carload and had found that the volatile matter did not burn off readily. The effect of this light charge is shown in the

platted curve of gas yield (Pl. IV), which drops decidedly after the first half hour.

In test 30 when a regular charge, 400 pounds, was used, the curve of gas yield is more like those platted from the tests of other coals. The coke was 69.4 per cent of the weight of the light charge, and 64.5 per cent of the heavier charge. The difference, though rather large, is in the logical direction. The coke was of good quality, only 9 per cent being classified as breeze.

The yield of gas in the first test was 4.81 cubic feet per pound of coal. The average candlepower was 14.5 and the average heating value 650 British thermal units. In the second test the yield was 5 cubic feet per pound of coal, the average candlepower was 14.1, and the heating value was 622 British thermal units per cubic foot. The figures for candle-feet per pound of coal (69.7 and 70.5) agree well, as do those for the heating value of the gas per pound of coal (3,126 and 3,110 B. t. u.).

The quantities of ammonia liquor in the two tests were about the same but the runs of tar differed, the larger charge giving decidedly more. The ammonia yield was low in both tests, 4.02 and 4.07 pounds per ton. The recovery was about 15 per cent of the theoretical—rather below the average. Naphthalene was determined only in test 30; the quantity was small. Hydrogen sulphide was determined in test 30 in the unpurified gas leaving the ammonia scrubber; the percentage was practically the same as for the Pittsburg coal, which averaged rather higher in sulphur. The weight balance shows an unaccounted-for loss of 5.6 per cent in one test and 7.7 per cent in the other.

The tests are concordant, and seem reliable. The coal is reported to have been used commercially as a gas coal. It promises well.

COAL A. A. 9, FROM SAGINAW, MICH.

The sample consisted of 43 sacks shipped from the Barnard mine, working the Saginaw bed, at Saginaw, Mich., on the Michigan Central Railroad. Perry Barker, of the United States Geological Survey, collected the mine samples and saw to the shipment of the coal tested. The data relating to this coal are to be found in Tables 3 to 8. Only one test (34) was made.

The coal had not been badly broken in shipment; the lumps showed some slate and a noticeable amount of pyrite. On screening, 9.7 per cent of the material passed through the $\frac{1}{4}$ -inch bar screen. The coal as received held 9.28 per cent of moisture and even after air drying carried 4.71 per cent. The ratio of volatile matter to fixed carbon was higher than in the Pittsburg coal. The ratio of oxygen to hydrogen was 1.71, compared with 1.28 for the Pittsburg coal.

A 400-pound charge of screened coal was put into a retort having a temperature decidedly below the average. The interior temperature of the retort remained low throughout the test; hence the yield of gas started slowly and did not reach its maximum until the sixth half hour. The yield of coke was 59.5 per cent; the coke contained 17.9 per cent breeze. The output of gas was low, only 4.3 cubic feet per pound of coal, but the average candlepower of the gas was 17.4, so that the candle feet were 74.8. On the other hand, the average heating value was only 593 British thermal units per cubic foot. The maximum rate of gas production coming at the middle of the test when the heat value of the gas had dropped below 600 British thermal units was one cause of this low average heat value, though even at the beginning the heat value was not very high. The combination of low yield and heating value gave the gas the extremely low average heat value of 2,550 British thermal units per pound of coal.

The quantity of ammonia liquor was large, a logical result of the high moisture content of the coal. The run of tar was about normal. The yield of ammonia was high, 6.40 pounds on the coal as charged, and 21.1 per cent of the theoretical yield. The high recovery probably resulted from the low retort temperature, which lessened the decomposition of the ammonia. The quantity of naphthalene was small. The proportion of hydrogen sulphide at the outlet of the scrubber was very large, 3,750 grains per cubic foot, or more than four times as much as from the Pittsburg coal, which contained about the same percentage of sulphur. The weight balance shows the unaccounted-for loss to be 8.5 per cent, or a little more than the average for all tests, 6.6 per cent.

The results throughout are concordant and the test seems reliable. This coal does not promise well as a gas coal, but it must be remembered that the low retort temperature had an unfavorable effect.

COAL A. A. 2, FROM VAN HOUTEN, N. MEX.

The sample consisted of 58 sacks of run-of-mine coal from the Raton bed, at Van Houten, Colfax County, N. Mex., on the Santa Fe Railroad. The mine sampling and the shipment of the sacks were under the supervision of K. M. Way. The usual analyses and the results of tests 22 and 28 are given in Tables 3 to 8.

The coal as received was rather fine, possibly because of the handling of the sacks in transit, but was clean, with a good luster. It showed little slate and no pyrite. Results of screening tests differed decidedly; in one test 11.95 per cent was rejected and in the other 21.90 per cent. The analyses of the two samples of coal gave similar discrepancies in the ash content, 15.1 per cent in test 22 and 8.94 per cent in test 28; the differences were corroborated by the ash content

of the coke, 22.6 per cent in test 22 and 17.66 per cent in test 28. The ash in the mine samples fell between the extremes shown by the samples taken at the plant, being 10 per cent and 12.7 per cent, respectively. The composition of the coal, aside from the ash, was not much unlike that of the Pittsburgh coal.

The retort temperatures were moderately high. The yield of coke was about the same as from the Pittsburgh coal. The proportion of breeze was small, averaging 12.5 for the two tests. The curves showing the gas yield and the variation in the composition of the gas (see Plate IV) are almost identical for the two tests. The candle-power of the gas (16.5 and 16.8) and its heat value (674 and 682 B. t. u.) agree well in the two tests.

The yield of gas was less on test 22 (4.5 against 4.9 cubic feet). On eliminating the effect of the higher ash in the coal of test 22 by calculating the yield to an ash and moisture free basis, the discrepancies disappear and the yield becomes 5.5 cubic feet per pound of coal in each case.

The runs of ammoniacal liquor and of tar are higher in test 28 than in test 22, and as the total products in test 28 add up to more than the weight of the coal charged, it is possible that the recorded weight of coal charged is wrong and should have been about 10 pounds higher. The yield of ammonia was larger in test 22 (4.98 pounds per ton) than in test 28 (4.34 pounds per ton). The loss in test 28 came in the last three half hours, where scarcely any ammonia was evolved. There may have been an error here, although there is nothing in the data to prove it. The weight balance shows an unaccounted-for loss of 5 per cent in test 22 and a negative loss of 0.3 per cent in test 28, as mentioned above. The fact that the yields of coke, gas, tar, and ammoniacal liquor are all higher in test 28 than in test 22 is another indication that the reported weight of the coal charged into the retort in test 28 was too low.

This coal in the tests gave as good results as Pittsburgh coal, and well deserves further investigation by gas companies in the Southwest.

COAL A. A. 10, FROM LA FOLLETTE, TENN.

The sample consisted of 60 sacks of coal from the Rex bed, at La Follette, Campbell County, Tenn., on the Louisville & Nashville Railroad. P. M. Riefkin inspected the mine and supervised the shipment of the sample. The composition of the coal and the results of test 31 are given in Tables 3 to 8.

The coal as received was bright and clean with no pyrite or slate evident. Only 8.37 per cent of the shipment passed through the $\frac{1}{4}$ -inch bar screen. The coal as sampled and analyzed at the plant showed a low moisture content and the extremely low ash content of

2.15 per cent, though the mine samples contained 5 and 7 per cent of ash, respectively. Except for the ash the proximate analysis of the coal is not very different from that of the Pittsburg coal, but the ultimate analysis shows 20 per cent more oxygen and a high percentage of nitrogen.

The usual charge, 400 pounds, of screened coal was placed in a moderately hot retort. The percentage of coke and of screenings were about the same as for the Pittsburg coal, although the coke seemed more fragile. The evolution of gas was regular. The total yield of gas was high, 5.5 cubic feet per pound of coal as charged, but the yield if calculated to a basis of coal ash and moisture free is only slightly higher than the average results from the Pittsburg coal. The candlepower (15.9) and heat value (641 B. t. u.) of the gas are also fairly comparable with the results from the latter coal. The platted curves showing the evolution of ammoniacal liquor and tar (Pl. IV) are regular. The yield of ammonia was high (7.39 per cent), largely from the high nitrogen content of the coal; the actual yield was only 17.9 per cent of the theoretical while the average proportion for all the tests was 16.8 per cent. No naphthalene tests were made. The percentage of hydrogen sulphide in the unpurified gas at the outlet of the scrubber was low. There may be some relation between it and the high percentage of ammonia, because of the removal of hydrogen sulphide by ammonia in the scrubber. The weight balance shows the rather low unaccounted-for loss of 2.1 per cent.

The test seems reliable and the coal deserves further examination.

COAL A. A. 5, FROM KANAWHA NO. 2 BED, PAGE, W. VA.

The sample consisted of 67 sacks of run-of-mine coal from the Kanawha No. 2 bed at Page, Fayette County, W. Va., on the Virginia Railway. G. S. Pope inspected the mine and shipped the sample. Tables 3 to 8 give the analyses of the samples and the results of tests 25 and 27.

The coal as received was in fine lumps, fairly clean and with no slate or pyrite evident. Screening ($\frac{3}{4}$ -inch screen) removed a large proportion of the coal, 63.27 per cent in one trial and 70 per cent in the other, but the composition of the screenings and of the screened coal differed very little. The average ash content was 6.20 per cent in the coal passing over the screen, and 6.57 per cent in that falling through. This coal had a low percentage of volatile matter and a notably low content of oxygen, the lowest found in any of the coals tested.

Two distillation tests were made, the retort being hot, and the conditions nearly the same in each. The yield of coke was high, a result from the low percentage of volatile matter in the coal, but was distinctly too high in test 25, the figure being 75.5 per cent, as

against 71.25 per cent in test 27. The sum of the products of distillation in test 25 is 101.7 per cent of the weight of coal charged and the yields were uniformly higher, with the single exception of the ammonia liquor, in test 25 than in test 27. The obvious explanation is that the actual weight of the coal charged was more than the recorded weight. The stated yields of gas (5.20 and 5.5 cubic feet per pound), the candlepower (15.8 and 16.1), the candle-feet (82.2 and 80.5), and the heat values (617 and 622 B. t. u. per cubic foot, and 3,208 and 3,110 B. t. u. per pound of coal), check fairly well. They would agree better if the stated weight of coal charged in test 25 were correct. The yield of ammonia was noticeably higher in test 25 (5.19 pounds per ton) than in test 27 (4.38 pounds per ton). The apparent error in the weight of coal charged may partly account for the difference.

Naphthalene was determined in test 27 and the result was the highest of any of the coals. Hydrogen sulphide in the unpurified gas at the outlet of the scrubber was determined in one test; it was twice as much as for the Pittsburg coal, though the latter did not contain a correspondingly higher percentage of sulphur. The weight balance of test 27 was about normal.

These tests tend to confirm the apparently prevalent opinion that West Virginia coals in general give good gas but yield much naphthalene. Another opinion that they produce a pitchy tar which stops standpipes, could not be tested in the short runs made. Although the yield of naphthalene was large, the quantity in the gas at the outlet of the tar separator was not larger than in one of the tests with Pittsburg coal.

COAL A. A. 7, FROM HANNA, WYO.

The sample consisted of 59 sacks of coal from the No. 2 (or lower) bed at Hanna, Carbon County, Wyo., on the Union Pacific Railroad. The sacks were filled from the railway cars as they left the tipple. K. M. Way inspected the mine and supervised the loading of the sample. The usual test data are given in Tables 3 to 8. The coal stood shipment well, though somewhat dusty when received; it showed no pyrite or slate. Only 10.4 per cent of the coal went through the $\frac{3}{4}$ -inch bar screen.

This coal was selected from a desire to test some of the high oxygen coals from the West, and this one with its 32.26 per cent of volatile matter seemed to be promising. The fact that the volatile matter was accompanied by 22 per cent of moisture was overlooked. The tests gave results of scientific interest, although the coal can not be considered in any sense a gas coal. The contained volatile matter figured to an ash and moisture free basis amounted to 44.04 per cent. The oxygen content was 18.93 per cent, and the ratio of oxygen to

hydrogen in this coal was 3.50 as compared with 1.28 in the Pittsburgh coal.

Two tests were made on this coal, the first one (No. 23) resulting badly. During the test, owing to a leak in the water line, the seals were blown twice, allowing the purifier to fill with water. Before the leak could be remedied the recorded yield of gas became hopelessly inaccurate. When the retort was opened at the end of the run, the coke was so fine that it flew in sparks all over the retort house. A later test on this coal was made to get more accurate data, but no attempt was made to take a full set of observations.

Test 29 gave the following results: Coke, 50.2 per cent; gas, 5.4 cubic feet per pound of coal; candlepower of gas, 8.8; heat value of gas, 564 British thermal units per cubic foot. The determinations of candlepower and heat value were on gas taken from the proportional tank, and the gas in this tank tends to give rather high results. The ammonia yield was low, 3.77 pounds per ton of coal. The weight balance showed a very high unaccounted-for loss, 14.3 per cent. This loss is probably due in part to coke lost in drawing, and in part to the high oxygen content of the coal causing the formation of much carbon dioxide which was lost in the ammonia scrubber. Even so there was 7.6 per cent of carbon dioxide in the purified gas.

TABULATED RESULTS.

TABLE 3.—Comparative analyses of coal and coke.

Test number and character of fuel.	Proximate analysis.					Heat value (B. t. u.).	Ultimate analysis.					
	Moisture.	Ash.	Volatile matter.	Fixed carbon.	Sulphur.		Hydrogen.	Carbon.	Nitrogen.	Oxygen.	Sulphur.	Ash.
Test 18 (A. A. I.).												
Coal:												
As received.....	1.92	6.41	32.82	58.85	1.12	14,026						
Dry.....		6.54	33.46	60.00	1.14	14,301	5.24	79.00	1.38	6.72	1.14	6.52
Moisture and ash free.....			35.80	64.20	1.22	15,302	5.61	84.51	1.48	7.18	1.22	
Coke:												
As received.....	8.54	11.46	.97	79.03	.84	11,552						
Dry.....		12.53	1.06	86.41	.92	12,631						
Moisture and ash free.....			1.21	98.79	1.05	14,440						
Test 19 (A. A. I.).												
Coal:												
As received.....	2.18	7.53	32.96	57.33	1.43	13,815						
Dry.....		7.70	33.70	58.60	1.46	14,123						
Moisture and ash free.....			36.51	63.49	1.58	15,300						
Coke:												
As received.....	18.23	9.00	1.22	71.55	.69	10,543						
Dry.....		11.01	1.49	87.50	.84	12,893						
Moisture and ash free.....			1.67	98.33	.94	14,488						
Test 20 (A. A. I.).												
Coal:												
As received.....	2.05	6.68	33.25	58.02	1.49	13,955						
Dry.....		6.82	33.94	59.24	1.52	14,247						
Moisture and ash free.....			36.42	63.58	1.63	15,289						
Coke:												
As received.....	27.97	8.17	.64	63.22	.80	9,207						
Dry.....		11.34	.89	87.77	1.11	12,782						
Moisture and ash free.....			1.00	99.00	1.25	14,416						

TABLE 3.—Comparative analyses of coal and coke—Continued.

Test number and character of fuel.	Proximate analysis.					Heat value (B. t. u.).	Ultimate analysis.					
	Moisture.	Ash.	Volatile matter.	Fixed carbon.	Sulphur.		Hydrogen.	Carbon.	Nitrogen.	Oxygen.	Sulphur.	Ash.
Test 21 (A. A. 3).												
Coal:												
As received	2.46	6.25	31.18	60.11	.43	13,885						
Dry		6.41	31.97	61.62	.44	14,234	5.01	79.73	1.22	7.56	.55	5.93
Moisture and ash free			34.16	65.84	.47	15,210	5.32	84.75	1.30	8.05	.58	
Coke:												
As received	12.43	10.09	.92	76.56	.36	11,210						
Dry		11.52	1.05	87.43	.41	12,802						
Moisture and ash free			1.19	98.81	.46	14,468						
Screenings:												
As received	4.30	12.67	26.88	56.15	.46	12,578						
Dry		13.24	28.09	58.67	.48	13,144						
Moisture and ash free			32.38	67.62	.55	15,149						
Test 22 (A. A. 2).												
Coal:												
As received	2.23	15.10	31.42	51.25	.67	12,438						
Dry		15.44	32.14	52.42	.69	12,722	5.08	73.68	1.28	6.01	.22	12.73
Moisture and ash free			38.01	61.99	.82	15,046	5.82	84.43	1.47	6.88	1.40	
Coke:												
As received	22.30	17.56	.58	59.56	.31	8,681						
Dry		22.60	.75	76.65	.40	11,173						
Moisture and ash free97	99.03	.52	14,434						
Screenings:												
As received	2.48	14.78	32.01	50.73	.70	12,335						
Dry		15.16	32.82	52.02	.72	12,648						
Moisture and ash free			38.68	61.32	.85	14,909						
Test 24 (A. A. 4).												
Coal:												
As received	7.17	14.55	32.36	45.92	1.00	10,953						
Dry		15.67	34.86	49.47	1.08	11,799	4.66	70.41	1.59	12.44	1.58	9.32
Moisture and ash free			41.34	58.66	1.28	13,991	5.14	77.65	1.75	13.72	1.74	
Coke:												
As received	21.39	19.93	1.40	57.28	.68	8,417						
Dry		25.35	1.78	72.87	.87	10,706						
Moisture and ash free			2.38	97.62	1.17	14,342						
Screenings:												
As received	6.79	6.50	34.52	52.19	1.52	12,272						
Dry		6.97	37.03	56.00	1.63	13,165						
Moisture and ash free			39.80	60.20	1.75	14,152						
Test 25 (A. A. 5).												
Coal:												
As received	1.28	5.57	30.65	62.50	1.53	14,531						
Dry		5.64	31.05	63.31	1.55	14,720	5.09	82.46	1.42	4.95	.83	5.25
Moisture and ash free			32.91	67.09	1.64	15,601	5.37	87.03	1.50	5.22	.88	
Coke:												
As received	22.59	9.37	.38	67.66	.66	9,855						
Dry		12.10	.49	87.41	.85	12,731						
Moisture and ash free56	99.44	.97	14,485						
Screenings:												
As received	1.50	7.71	29.45	61.34	1.45	14,161						
Dry		7.83	29.90	62.27	1.47	14,377						
Moisture and ash free			32.44	67.56	1.59	15,599						
Test 28 (A. A. 2).												
Coal:												
As received	2.32	8.94	32.18	56.56	.70	13,385						
Dry		9.15	32.95	57.90	.72	13,703						
Moisture and ash free			36.27	63.73	.79	15,082						
Coke:												
As received												
Dry												
Moisture and ash free												
Screenings:												
As received	2.22	9.47	34.31	54.00	.51	13,318						
Dry		9.68	35.09	55.23	.52	13,621						
Moisture and ash free			38.85	61.15	.58	15,080						

TABLE 3.—Comparative analyses of coal and coke—Continued.

Test number and character of fuel.	Proximate analysis.					Ultimate analysis.					
	Moisture.	Ash.	Volatile matter.	Fixed carbon.	Sulphur.	Heat value (B. t. u.).	Hydrogen.	Carbon.	Nitrogen.	Oxygen.	Sulphur.
<i>Test 29 (A. A. 7).</i>											
Coal:											
As received.....	22.56	5.81	32.28	39.37	.36	9,592					
Dry.....		7.50	41.66	50.84	.46	12,386	5.03	68.80	1.08	17.58	.42
Moisture and ash free.....			45.04	54.96	.50	13,390	5.41	78.05	1.16	18.93	.45
Coke:											
As received.....	16.25	11.49	1.06	71.20	.34	10,447					
Dry.....		13.72	1.27	85.01	.41	12,474					
Moisture and ash free.....			1.47	98.53	.48	14,458					
Screenings:											
As received.....	11.45	6.13	37.06	45.36	.32	10,939					
Dry.....		6.92	41.85	51.23	.36	12,353					
Moisture and ash free.....			44.96	55.04	.39	13,271					
<i>Test 26 (A. A. 6).</i>											
Coal:											
As received.....	.71	18.32	24.14	56.83	.57	12,436					
Dry.....		18.45	24.31	57.24	.58	12,524	4.48	72.83	1.07	6.46	.66
Moisture and ash free.....			29.81	70.19	.71	15,358	5.24	85.18	1.25	7.56	.77
Coke:											
As received.....	23.09	15.89	.73	60.29	.38	8,998					
Dry.....		20.66	.95	78.39	.49	11,700					
Moisture and ash free.....			1.20	98.80	.62	14,747					
Screenings:											
As received.....	1.45	17.96	24.44	56.13	.56	12,407					
Dry.....		18.24	24.80	56.96	.57	12,589					
Moisture and ash free.....			30.33	69.67	.70	15,397					
<i>Test 31 (A. A. 10).</i>											
Coal:											
As received.....	3.13	2.15	34.99	59.73	.72	14,245					
Dry.....		2.22	36.12	61.66	.74	14,706	5.00	78.14	1.67	8.30	1.83
Moisture and ash free.....			36.94	63.06	.76	15,039	5.27	82.30	1.76	8.74	1.93
Coke:											
As received.....	26.80	4.45	.86	67.89	.51	10,094					
Dry.....		6.08	1.17	92.75	.70	13,790					
Moisture and ash free.....			1.26	98.75	.75	14,683					
Screenings:											
As received.....	3.21	5.21	33.99	57.59	1.21	13,750					
Dry.....		5.38	35.12	59.50	1.25	14,207					
Moisture and ash free.....			37.12	62.88	1.32	15,016					

TABLE 4.—Analyses and composition of coals.

COAL AS CHARGED.

Test No.—	Proximate analysis.					Heat value (B. t. u.).
	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	
33.....	2.71	29.13	63.87	4.29	.50	13,990
24.....	7.17	32.36	45.92	14.55	1.00	10,953
26.....	1.70	23.90	56.26	18.14	.57	12,312
35.....	4.66	34.44	53.71	7.19	1.96	12,919
21.....	2.46	31.18	60.11	6.25	.43	13,885
30.....	3.17	32.40	60.67	3.76	.45	14,200
34.....	9.28	31.67	53.70	5.35	.98	12,456
22.....	2.23	31.42	51.25	15.10	.67	12,438
28.....	2.32	32.18	56.56	8.94	.70	13,385
18.....	1.92	32.82	58.85	6.41	1.12	14,026
19.....	2.18	32.96	57.33	7.53	1.43	13,815
20.....	2.05	33.25	58.02	6.68	1.49	13,955
32.....	2.43	32.70	59.99	4.88	.85	14,036
31.....	3.13	34.99	59.73	2.15	.72	14,245
25.....	1.28	30.65	62.50	5.57	1.53	14,531
27.....	1.27	29.02	62.88	6.83	1.18	14,344
29.....	22.56	32.26	39.37	5.81	.36	9,592

TABLE 4.—*Analyses and composition of coals*—Continued.

DRY COAL.

Test No.	Calculated proximate composition.				Heat value (B. t. u.)	Ultimate analysis. ^a					
	Volatile matter.	Fixed carbon.	Ash.	Sulphur.		Hydrogen.	Carbon.	Nitrogen.	Oxygen.	Sulphur.	Ash.
33.....	29.94	65.65	4.41	0.51	14,380	5.35	80.50	1.24	8.21	0.62	4.08
24.....	34.86	49.47	15.67	1.08	11,799	4.66	70.41	1.59	12.44	1.58	9.32
26.....	24.31	57.24	18.46	.58	12,524	4.48	72.83	1.07	6.46	.66	14.50
35.....	36.13	56.33	7.54	2.06	13,550	4.89	76.21	1.43	7.93	1.77	7.77
21.....	31.97	61.62	6.41	.44	14,234	5.01	79.73	1.22	7.56	.55	5.93
30.....	33.46	62.66	3.88	.46	14,065
34.....	34.91	59.19	5.90	1.08	13,730	5.13	75.56	1.35	8.79	1.41	7.76
22.....	32.14	52.42	15.44	.69	12,722	5.08	73.68	1.28	6.01	1.22	12.73
28.....	32.95	57.90	9.15	.92	13,703
18.....	33.46	60.00	6.54	1.14	14,301	5.24	79.00	1.38	6.72	1.14	6.52
19.....	33.70	58.80	7.70	1.46	14,123
20.....	33.94	59.24	6.82	1.52	14,247
32.....	33.51	61.49	5.00	.87	14,386
31.....	36.12	61.66	2.22	.74	14,706	5.00	78.14	1.67	8.30	1.83	5.06
25.....	21.06	63.31	5.64	1.55	14,720	5.09	82.46	1.42	4.95	.83	5.25
27.....	29.39	63.69	6.92	1.20	14,330
29.....	41.66	50.84	7.50	.46	12,386	5.03	68.80	1.08	17.58	.42	7.09

^a The ultimate analyses are mostly of the mine samples, and hence differ from analyses of test samples, especially in the percentages of ash. The composition of the coal free from moisture and ash probably does not differ much.

COMPOSITION OF COALS.

Test No.	Calculated proximate composition.			Heat value (B. t. u.)	Calculated ultimate composition.				
	Volatile matter.	Fixed carbon.	Sulphur.		Hydrogen.	Carbon.	Nitrogen.	Oxygen.	Sulphur.
33.....	31.32	66.68	0.53	15,043	5.58	83.92	1.29	8.56	0.65
24.....	41.34	58.66	1.28	13,991	5.14	77.65	1.75	13.72	1.74
26.....	28.81	70.19	.71	15,358	5.24	85.18	1.25	7.56	.77
35.....	39.07	60.93	2.23	14,656	5.30	82.63	7.55	8.60	1.92
21.....	34.16	65.84	.47	15,210	5.32	84.75	1.30	8.05	.58
30.....	34.81	65.19	.48	15,257
34.....	37.10	62.90	1.15	14,591	5.56	81.91	1.46	9.54	1.53
22.....	38.01	61.99	.82	15,046	5.82	84.43	1.47	6.88	1.40
28.....	36.27	63.73	.79	15,082
18.....	35.80	64.20	1.22	15,302	5.61	84.51	1.48	7.18	1.22
19.....	36.51	63.49	1.58	15,300
20.....	36.42	63.58	1.63	15,289
32.....	35.27	64.73	.92	15,192
31.....	36.94	63.06	.76	15,039	5.27	82.30	1.76	8.74	1.93
25.....	32.91	67.09	1.64	15,601	5.37	87.03	1.50	5.22	.88
27.....	31.57	68.43	1.29	15,610
29.....	45.04	54.96	.50	13,390	5.41	78.05	1.16	18.93	.45

TABLE 5—Retort operation, coal and coke.

Test No.	Coal.						Coke.				
	Coal No.	State.	Per cent re- jected by 1/2-inch bar screen.	Screened, as charged (pounds).	Charged, dry (pounds).	Charged, ash and moist- ure free (pounds).	Yield (pounds).	Dry (per cent of coal charged).	Dry (per cent of dry coal charged).	Per cent of breaze through 1/2- inch bar screen.	
33.....	A. A. 8...	Alabama.....	23.40	400	389	372	273	68.3	70.2	12.5	
24.....	A. A. 4...	Colorado.....	39.50	400	371	313	240	60.0	64.6	37.3	
26.....	A. A. 6...	do.....	13.83	410	403	339	307	74.9	76.0	10.0	
35.....	A. A. 12...	Illinois.....	27.90	400	381	353	249	62.3	65.4	6.7	
21.....	A. A. 3...	Kentucky.....	^a 30.80	360	351	329	250	69.4	71.3	
30.....	do.....	do.....	48.50	400	388	372	258	64.5	66.5	9.1	
34.....	A. A. 9...	Michigan.....	9.70	400	363	341	238	59.5	65.6	17.9	
22.....	A. A. 2...	New Mexico.....	11.95	398	389	329	270	67.8	69.4	13.1	
28.....	A. A. 1...	do.....	21.90	400	391	355	280	70.0	71.0	11.7	
18.....	do.....	Pennsylvania.....	(b)	404	396	320	271	67.1	68.5	5.0	
19.....	do.....	do.....	(b)	400	391	370	270	67.5	69.0	
20.....	do.....	do.....	(b)	400	392	365	259	64.8	66.1	
32.....	do.....	do.....	17.40	400	390	370	274	68.5	70.2	7.6	
31.....	A. A. 10...	Tennessee.....	8.37	400	387	379	267	66.8	69.0	7.6	
25.....	A. A. 5...	West Virginia.....	63.27	400	395	372	302	75.5	76.5	10.0	
27.....	do.....	do.....	70.00	400	394	368	285	71.2	72.4	7.6	
29.....	A. A. 7...	Wyoming.....	10.4	400	309	296	201	50.2	64.9	88.4	

Test No.	Products of destructive distillation.						Heat value of coke (B.t.u.)	Carbonization period (hours).	Retort temperature (°F.).		
	Coal No.	State.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.			On top of coal at center of retort.		Average at exterior.
									After 1 hour.	Maximum.	
33.....	A. A. 8...	Alabama.....	1.59	81.01	11.40	0.52	12,883	4.63	1,240	1,345	1,878
24.....	A. A. 4...	Colorado.....	1.78	72.87	25.35	.87	10,706	4.50	1,320	1,700	2,036
26.....	A. A. 6...	do.....	.95	78.39	20.66	.49	11,700	4.42	1,430	1,820	2,123
35.....	A. A. 12...	Illinois.....	2.22	85.48	12.30	2.08	12,427	4.67	1,220	1,360	1,779
21.....	A. A. 3...	Kentucky.....	1.05	87.43	11.52	.41	12,802	4.42	1,470	1,740	2,036
30.....	do.....	do.....	.39	89.60	10.01	.43	13,003	4.50	1,340	1,550	2,020
34.....	A. A. 9...	Michigan.....	2.20	92.01	5.79	.91	13,495	4.75	1,090	1,140	1,828
22.....	A. A. 2...	New Mexico.....	.75	76.65	22.60	.40	11,173	4.48	1,390	1,690
28.....	A. A. 1...	do.....	2.75	79.69	17.56	.54	11,781	4.42	1,748	1,962
18.....	do.....	Pennsylvania.....	1.06	86.41	12.53	.92	12,631	4.48	1,770	1,982
19.....	do.....	do.....	1.49	87.50	11.01	.84	12,893	4.83	1,350	1,690
20.....	do.....	do.....	.89	87.77	11.84	1.11	12,782	4.55	1,460	1,755	2,010
32.....	do.....	do.....	1.85	89.45	8.70	.74	13,086	4.85	1,810	1,400	1,844
31.....	A. A. 10...	Tennessee.....	1.17	92.75	6.08	.70	13,790	4.67	1,810	1,465	1,938
25.....	A. A. 5...	West Virginia.....	.49	87.41	12.10	.85	12,731	4.52	1,397	1,710	2,042
27.....	do.....	do.....	.73	86.01	13.26	1.19	12,591	4.50	1,230	1,725	2,101
29.....	A. A. 7...	Wyoming.....	1.27	85.01	13.72	.41	12,474	4.25

^a A ½-inch bar screen was used in this test.^b The coal in these tests had been passed over a ½-inch bar screen at the mine and was not screened again.

TABLE 6.—*Yield, heating value, and candlepower of gas.*

Test No.	Yield of gas per pound of coal (cubic feet).			Calculated average heating value (B. t. u.).		Calculated average candlepower.	Calculated average gas analysis (per cent).						
	Coal as charged.	Dry coal.	Coal ash and moisture free.	Gross.	Net.		CO ₂ .	CuH ₂ O.	O ₂ .	CO.	CH ₄ .	H ₂ .	N ₂ .
33.....	5.2	5.3	5.6	598	538	14.0	2.7	3.7	1.0	8.9	31.2	49.1	3.4
24.....	4.8	5.2	6.1	a 626	a 566	a 14.0	a 4.4	a 5.3	a 0.8	a 11.3	a 29.8	a 43.9	a 4.5
26.....	4.9	5.0	6.1	614	550	a 12.0	1.3	a 4.2	0.9	6.0	29.5	55.8	2.3
35.....	4.3	4.6	4.9	a 632	a 568	15.2	2.5	3.7	0.8	7.9	33.5	45.6	6.0
21.....	4.8	4.9	5.3	650	578	14.5	a 1.6	a 4.3	a 1.1	a 6.4	a 33.1	a 50.3	a 3.2
30.....	5.0	5.2	5.4	622	526	14.1	1.9	4.1	0.7	7.0	29.4	51.3	5.6
34.....	4.3	4.7	5.5	593	526	17.4	a 2.9	a 5.3	a 0.7	a 9.2	a 34.1	a 43.1	a 4.7
22.....	4.5	4.7	5.5	674	606	a 16.5	2.0	a 5.9	0.8	8.2	a 34.8	a 45.9	2.4
28.....	4.9	5.0	5.5	682	618	16.8	2.1	6.1	0.6	8.2	35.5	45.7	1.8
18.....	4.8	4.9	6.1
19.....	4.9	5.0	5.3	649	598	a 15.2	1.2	4.4	1.1	6.6	a 37.9	a 42.2	6.6
20.....	5.3	5.4	5.8	611	550	16.0	1.1	3.0	0.8	6.5	31.8	51.6	5.2
32.....	5.1	5.3	5.6	a 604	a 603	a 15.4	a 1.7	a 3.8	a 0.5	a 8.5	a 33.6	a 50.7	a 3.2
31.....	5.5	5.7	5.8	641	575	15.9	2.0	5.0	0.9	8.2	31.2	49.1	3.6
25.....	5.2	5.3	5.6	617	555	a 15.8	1.3	3.8	0.9	4.9	a 32.9	a 53.6	2.6
27.....	5.0	5.1	5.4	622	557	a 16.1	1.2	3.3	0.6	5.0	32.6	55.1	1.8
29.....	5.4	7.0	7.5	b 564	b 502	8.8	b 7.6	b 4.9	b 0.2	b 14.4	b 29.0	b 40.2	b 3.7

a One figure of the nine used in obtaining the average was supplied by interpolation.

b From the proportional tank; no half-hour samples were taken on this test.

TABLE 7.—*Yield of tar, ammonia, sulphur, and naphthalene.*

Test No.	Tar per ton of coal (gallons).			Ammonia (NH ₃) per ton of coal (pounds).			H ₂ S per 100 cubic feet of gas at outlet of scrubber (grains).	Total naphthalene per ton of coal as charged (pounds).
	Coal as charged.	Dry coal.	Coal ash and moisture free.	Coal as charged.	Dry coal.	Coal ash and moisture free.		
33.....	19.2	19.7	20.6	4.03	4.15	4.33	10.7
24.....	15.6	16.8	19.9	7.60	8.20	9.60
26.....	14.2	14.5	17.7	4.36	4.43	5.43
35.....	17.2	18.1	19.5	6.38	6.70	7.20	1,300	4.5
21.....	10.8	11.1	11.8	4.01	4.12	4.39
30.....	18.8	19.5	20.2	3.48	3.61	4.25	823	5.7
34.....	16.5	18.2	19.3	a 6.40	a 7.06	a 7.50	3,750	5.5
22.....	16.8	17.2	20.3	a 4.97	a 5.09	a 6.02
28.....	21.8	22.3	24.5	4.32	4.42	4.82
18.....
19.....	10.7	10.9	11.8	5.80	5.93	6.43
20.....	5.18	5.28	5.67
32.....	21.1	21.6	22.8	5.33	5.46	5.75	830	8.0
31.....	22.0	22.7	23.2	a 7.39	a 7.63	a 7.81	296
25.....	20.1	20.4	21.7	5.19	5.26	5.57
27.....	14.6	14.8	15.9	4.37	4.43	4.75	11.4
29.....	14.4	18.6	20.1	3.77	4.88	5.28	1,660

a One figure of the nine used in obtaining the average was supplied by interpolation.

TAB' 8.—Results of tests on eleven coals from various States.

Source.	Coal A. A. No.	Test No.	Retort temperatures (°F.).		Exterior, average.	Average temperatures of condensing system (°F.).			Gas leaving station meter.		
			Interior.			Condenser.				Tar extractor.	
			After 1 hour.	Maxi- mum.		No. 1.	No. 2.	No. 3.		Inlet.	Outlet.
Scott Haven, Pa.	1	18	1,350	1,770	1,994	160	143	123	116	105	53
		19	1,690	1,980	1,994	127	124	126	107	100	60
	8	20	1,460	1,755	2,010	151	133	129	105	100	53
		21	1,400	1,400	1,844	136	129	111	104	83	54
	4	22	1,310	1,700	1,878	144	145	125	106	103	61
		23	1,240	1,345	2,086	157	144	126	109	106	58
	6	24	1,320	1,700	2,086	110	107	104	98	86	57
		25	1,430	1,820	2,123	163	146	121	110	106	58
	12	26	1,220	1,360	1,779	138	146	121	110	106	70
		27	1,470	1,740	2,086	120	122	116	109	93	48
Blount, Ala.	3	28	1,340	1,550	2,020	137	122	117	109	83	63
		29	1,340	1,550	2,020	137	122	117	109	83	63
	9	30	1,470	1,640	1,828	174	158	138	127	124	70
		31	1,080	1,140	1,528	129	128	116	105	99	63
	22	32	1,380	1,690	1,993	126	128	116	105	99	64
		33	1,748	1,748	1,993	135	116	112	104	97	60
	28	34	1,310	1,465	1,838	138	137	114	106	95	58
		35	1,710	1,710	2,042	112	134	105	95	81	52
	10	36	1,397	1,705	2,042	112	134	105	95	81	52
		37	1,710	1,710	2,042	112	134	105	95	81	52
La Follette, Tenn.	5	38	1,280	1,728	2,101	116	124	114	104	92	54
		39	1,280	1,728	2,101	116	124	114	104	92	54
	27	40	1,280	1,728	2,101	116	124	114	104	92	54
		41	1,280	1,728	2,101	116	124	114	104	92	54
	7	42	1,280	1,728	2,101	116	124	114	104	92	54
		43	1,280	1,728	2,101	116	124	114	104	92	54
	7	44	1,280	1,728	2,101	116	124	114	104	92	54
		45	1,280	1,728	2,101	116	124	114	104	92	54
	7	46	1,280	1,728	2,101	116	124	114	104	92	54
		47	1,280	1,728	2,101	116	124	114	104	92	54

TABLE 8.—Results of individual tests, etc.—Continued.

Source.	Coal A. A. No.	Test No.	Coke (per cent of coal charged).	Per pound of coal charged (cubic feet, cor- rected).	Average candle- power (approx- imate).	Candle- feet per pound of coal.	Average heat- ing value per cubic foot (B. t. u.).		Average gas analysis (per cent).								
							Gross.	Net.	Heating value per pound of coal (B. t. u.).	Carbon dioxide.	Illumi- nants.	Oxygen.	Carbon monox- ide.	Methane.	Hydro- gen.	Nitro- gen.	
Scott Haven, Pa.	1	18	67.1	4.84	15.2	75.00	640	593	1.2	4.4	1.1	8.6	37.9	42.2	6.6		
		19	67.5	4.93	16.0	84.30	611	550	1.1	3.0	.8	6.5	31.8	51.6	5.2		
		20	64.8	5.27	15.4	70.20	664	503	1.7	3.8	.6	6.5	33.6	50.7	5.2		
		32	68.5	5.14	15.4	72.50	598	538	2.7	3.7	1.0	8.9	31.2	46.1	3.4		
		33	68.3	4.81	14.0	67.20	626	566	4.4	5.3	.8	11.3	29.8	43.9	4.5		
		4	69.9	4.90	12.0	58.80	614	550	1.3	4.2	.9	6.0	29.5	55.8	2.3		
		6	75.0	4.30	15.2	65.30	632	568	1.6	3.7	.8	7.9	33.5	45.6	6.0		
		12	62.3	4.81	14.5	69.75	650	578	2.5	4.3	1.1	6.4	33.1	50.3	3.2		
		3	21	69.4	5.00	14.1	70.50	622	561	1.9	4.1	.7	7.0	29.4	51.3	5.6	
		9	30	64.5	4.81	17.4	74.80	583	526	2.9	5.3	.7	9.2	34.1	43.1	4.7	
Bleaton, Ala.	8	34	59.5	4.30	16.5	82.30	674	606	2.0	6.9	.8	8.2	34.8	45.9	2.4		
		2	22	67.8	4.50	15.8	82.30	682	618	2.1	6.1	.6	8.2	35.5	45.7	1.8	
		28	70.0	4.90	16.8	81.90	641	575	2.0	5.0	.9	8.2	31.2	46.1	3.6		
		10	31	66.8	5.50	15.9	82.20	617	555	1.3	3.8	.9	4.9	32.9	53.6	2.6	
		5	25	75.5	5.20	15.8	80.50	622	557	1.2	3.3	.6	5.0	32.6	55.5	1.8	
		7	27	71.3	5.00	16.1	80.50	622	557	1.2	3.3	.6	5.0	32.6	55.5	1.8	
		29	50.2	5.40	8.8	47.50	564	502	7.6	4.9	.2	14.4	29.0	40.2	3.7		
		Bleaton, Ala.	8	32	68.5	5.14	15.4	72.50	598	538	2.7	3.7	1.0	8.9	31.2	46.1	3.4
				33	68.3	4.81	14.0	67.20	626	566	4.4	5.3	.8	11.3	29.8	43.9	4.5
				4	69.9	4.90	12.0	58.80	614	550	1.3	4.2	.9	6.0	29.5	55.8	2.3
6	75.0			4.30	15.2	65.30	632	568	1.6	3.7	.8	7.9	33.5	45.6	6.0		
12	62.3			4.81	14.5	69.75	650	578	2.5	4.3	1.1	6.4	33.1	50.3	3.2		
3	21			69.4	5.00	14.1	70.50	622	561	1.9	4.1	.7	7.0	29.4	51.3	5.6	
9	30			64.5	4.81	17.4	74.80	583	526	2.9	5.3	.7	9.2	34.1	43.1	4.7	
2	22			67.8	4.50	15.8	82.30	674	606	2.0	6.9	.8	8.2	34.8	45.9	2.4	
10	31			66.8	5.50	15.9	82.20	617	555	1.3	3.8	.9	4.9	32.9	53.6	2.6	
5	25			75.5	5.20	15.8	80.50	622	557	1.2	3.3	.6	5.0	32.6	55.5	1.8	
Hanna, Wyo.	7	27	71.3	5.00	16.1	80.50	622	557	1.2	3.3	.6	5.0	32.6	55.5	1.8		
		29	50.2	5.40	8.8	47.50	564	502	7.6	4.9	.2	14.4	29.0	40.2	3.7		

* One figure from the nine used in obtaining average was supplied by interpolation from the curves.

MANNER OF RECORDING AND COMPUTING DATA.

GENERAL STATEMENT.

The purpose of this bulletin is twofold—to present data obtained in tests made on a scale that would give results comparable with those of ordinary practice, and to show the methods used in an illuminating-gas experiment station that embodies the conclusions reached after much study and many trials.

In furtherance of both these purposes, specimens of the record sheets, forms for computing, and computation sheets used at the plant are reproduced herewith. The record and computation sheets are filled out for Test 35. They show all the data recorded and the methods used in getting the results stated in Tables 3 to 8 and platted in Plate IV. By going over them the reader can determine for himself the reliability of the general plan followed. The chief reasons for inserting the sheets, however, is that they may be of aid to any engineer who is preparing to make or is carrying on distillation tests of gas coals, by serving as guides in the preparation of forms and record blanks that will meet his needs.

SHEETS FOR RECORDING TESTS.

SHEET A 1.

Test No. 35.

COAL AND COKE.

Date: May 28, 1909.

Observer: D. F. Smith.

Coal:

Time of charging.....	12.40 p. m.
Weight of car and coal.....	Balanced.
Weight of car.....	
Weight of coal charged... pounds..	400

Kind of coal.....

Coal No..... A. A. 12

Company.....

Mine location..... Harrisburg, Saline County, Ill.

Analysis, as charged:

Moisture.....	4.66
Ash.....	7.19
Volatile matter.....	34.44
Fixed carbon.....	53.71

Sulphur, separately determined..... 1.96

Heating value..... B. t. u.. 12,919

Remarks:

Retort badly covered on top with carbon, temperature low.

Coke:

Time of drawing..... 5.20 p. m.

Weight of buggy and coke, pounds..... 636

Weight of buggy..... pounds.. 387

Weight of coke..... do.... 249

Coke produced..... per cent.. 62.3

Physical characteristics.. Dark color; easily broken.

Analysis, before quenching:

Moisture.....	
Ash.....	12.30
Volatile matter.....	2.22
Fixed carbon.....	85.48

Sulphur, separately determined..... 2.08

Heating value..... B. t. u.. 12,427

Remarks:

Sheets for recording tests—Continued.

SHEET A 2.
Test No. 35.Date: May 28, 1909.
Observer: Wynne.

RETORT TEMPERATURE.

Time of taking sample.	Distance from rear.	Temperature inside retort (° F.).			Time of taking sample.	Temperature outside retort (° F.).		
		At terminals.	Observed.	Corrected.		Distance from retort.	Observed.	Corrected.
<i>p. m.</i>					<i>p. m.</i>			
12.50	4 feet 6 inches		1,050	1,095	12.50	1½ inches	1,765	1,790
1.00	do.		1,115	1,150	1.00	do.	1,775	1,800
1.10	do.		1,140	1,175	1.10	do.	1,785	1,810
1.20	do.		1,160	1,185	1.20	do.	1,760	1,775
1.30	do.		1,160	1,190	1.30	do.	1,760	1,780
1.40	do.		1,175	1,200	1.40	do.	1,760	1,780
1.50	do.		1,190	1,220	1.50	do.	1,740	1,760
2.00	do.		1,205	1,230	2.00	do.	1,740	1,760
2.10	do.		1,215	1,240	2.10	do.	1,745	1,765
2.20	do.		1,220	1,250	2.20	do.	1,745	1,765
2.30	do.		1,240	1,260	2.30	do.	1,750	1,775
2.40	do.		1,265	1,285	2.40	do.	1,760	1,780
2.50	do.		1,270	1,290	2.50	do.	1,765	1,790
3.00	do.		1,295	1,310	3.00	do.	1,760	1,780
3.10	do.		1,325	1,340	3.10	do.	1,755	1,780
3.20	do.		1,335	1,350	3.20	do.	1,770	1,795
3.30	do.		1,335	1,350	3.30	do.	1,770	1,795
3.40	do.		1,345	1,355	3.40	do.	1,765	1,790
3.50	do.		1,350	1,360	3.50	do.	1,750	1,775
4.00	do.		1,345	1,355	4.00	do.	1,745	1,765
4.10	do.		1,340	1,350	4.10	do.	1,740	1,760
4.20	do.		1,335	1,350	4.20	do.	1,745	1,765
4.40			1,325	1,340			1,750	1,775
5.00			1,325	1,340			1,760	1,775
						Av.		1,779

SHEET A 3.
Test No. 35.Date: May 29, 1909.
Observer: D. F. Smith.

GENERAL TEMPERATURE.

Time of taking sample.	Condenser temperatures.			Separator temperatures.		Meter temperatures.		Purifier.
	1	2	3	Inlet.	Outlet.	Water.	Outlet.	
<i>p. m.</i>								
12.50	178	192	180	166	124	70	69
1.00	178	182	170	162	126	70	69
1.10	180	170	158	150	132	70	69
1.20	176	170	150	138	126	70	69
1.30	174	168	140	128	119	70	69
1.40	170	158	130	119	118	71	70
1.50	160	153	118	110	110	71	69
2.00	160	144	115	105	107	71	69
2.10				100	103	71	69
2.20				98	100	71	69
2.30	150	140	105	98	100	71	69
2.40				101	100	71	69
2.50				104	100	71	69
3.00	140	138	118	106	102	71	69
3.10				107	100	71	69
3.20				106	100	71	69
3.30	130	130	110	105	100	72	69
3.40				102	98	72	70
3.50				102	98	72	70
4.00	120	134	112	100	95	72	70
4.10				96	94	72	70
4.20				92	93	72	70
4.30	118	140	115	92	93	72	70
4.40				90	92	72	70
4.50	110	142	100	88	91	72	70
5.00							
Corrected average.....	162	146	121	110	105	72	70

Sheets for recording tests—Continued.

SHEET A 4.

Test No. 35.

Date: May 28, 1909.

Observer: D. F. Smith.

PRESSURES.

Time of taking sample.	Stand- pipe (inches).	Separator inlet (inches).	Separator difference (inches).	Washer difference (inches).	Meter (inches).	Meter reading.	
						Total. (cu. ft.).	Period. (cu. ft.).
<i>p. m.</i>							
12.40	0.1					00	
12.45						30	30
12.50	0.1	0.2	2.1	3.8	4.9	64	34
12.55	0.1	0.1	2.0	3.8	4.8	95	31
1.00						124	29
1.05	0.1	0.2	1.8	3.8	4.9	154	30
1.10	0.0	-0.1	1.9	3.8	4.7	185	31
1.15	-0.1	-0.1	1.9	3.8	4.7	216	31
1.20	0.2	0.2	1.9	3.8	4.5	248	32
1.25	0.0	0.0	1.9	3.6	4.7	279	31
1.30	0.1	0.1	1.9	3.2	4.7	313	34
1.35	0.3	0.3	1.9	3.0	4.8	346	33
1.40	0.0	0.1	1.9	3.0	5.0	380	34
1.45	0.0	0.0	1.8	3.0	5.0	414	34
1.50	0.0	0.0	1.8	3.0	5.0	448	34
1.55	-0.1	-0.1	1.8	3.0	5.0	482	34
2.00	0.0	0.0	1.7	3.0	5.0	518	36
2.05	0.2	0.2	1.7	3.0	5.0	553	35
2.10	0.1	0.1	1.7	3.0	5.0	590	37
2.15	0.2	0.2	1.7	3.0	5.0	627	37
2.20	0.2	0.2	1.7	3.0	5.0	660	33
2.25	0.1	0.1	1.7	3.0	5.0	702	42
2.30	-0.1	-0.1	1.7	3.0	5.0	740	38
2.35	-0.1	-0.1	1.7	3.0	5.0	779	39
2.40	0.0	0.0	1.7	3.0	5.0	818	39
2.45	0.0	0.0	1.7	3.1	5.0	858	40
2.50	0.1	0.1	1.7	3.0	5.0	899	41
2.55	0.1	0.1	2.0	3.0	5.0	939	40
3.00	0.1	0.1	1.9	3.0	5.0	980	41
3.05	0.0	-0.1	1.8	3.0	5.0	1,021	41
3.15	0.0	0.0	1.9	3.0	5.0	1,103	32
3.20	0.1	0.1	1.9	3.0	5.0	1,144	41
3.25	-0.1	-0.1	2.0	3.0	5.0	1,184	40
3.30	-0.1	-0.1	1.9	3.0	5.0	1,226	42
3.35	-0.1	-0.1	1.8	3.0	5.0	1,267	41
3.40	0.0	0.0	1.7	3.0	5.0	1,308	41
3.45	0.1	0.0	1.7	3.0	5.0	1,348	40
3.50	-0.1	-0.1	1.8	3.0	5.0	1,385	37
3.55	0.0	-0.1	1.8	2.9	5.0	1,428	43
4.00	0.0	0.0	1.8	3.0	5.0	1,466	38
4.05	0.1	0.1	1.7	3.0	5.0	1,503	37
4.10	0.1	0.1	1.7	2.0	5.0	1,540	37
4.15	0.1	0.1	1.7	2.0	5.0	1,573	33
4.20	0.0	-0.1	1.6	3.0	5.1	1,605	32
4.25	0.0	-0.2	1.6	2.0	5.0	1,636	31
4.30	0.1	0.1	1.5	3.0	5.0	1,665	29
4.35	0.1	0.0	1.5	3.0	5.0	1,691	26
4.45	0.1	0.1	1.5	3.0	5.0	1,740	49
4.50	0.1	0.1	0.8	3.0	5.0	1,760	20
4.55	0.0	0.0	0.8	3.0	5.0	1,776	16
5.00	0.0	0.0	0.7	3.0	5.0	1,788	12
5.05	0.0	0.0	0.5	3.0	5.0	1,799	11
5.10						1,807	8
						1,814	7

Sheets for recording tests—Continued.

SHEET A 5.
Test No. 35.Date: May 28, 1909.
Observer: Wynne.

CALORIMETER.

Time of taking sample.	Water inlet—temperature (°C.).						Water outlet—temperature (°C.).					
	1	2	3	4	5	Average.	1	2	3	4	5	Average.
<i>p. m.</i>												
12.45 to 1.15	16.1	16.1	16.1	16.1	16.1	16.1	42.0	41.9	41.9	42.0	42.1	41.98
1.15 to 1.45	15.4	15.4	15.4	15.4	15.4	15.4	42.9	43.0	43.2	43.3	43.5	43.18
1.45 to 2.15	15.6	15.6	15.7	15.7	15.7	15.66	41.9	41.9	42.0	42.1	42.2	42.02
2.15 to 2.45	16.0	16.0	16.0	16.1	16.1	16.04	45.1	45.1	45.1	45.0	45.1	45.08
2.45 to 3.15	15.9	16.0	16.0	16.0	16.1	16.0	46.5	46.6	46.7	46.8	46.9	46.6
3.15 to 3.45	16.1	16.2	16.2	16.1	16.1	16.14	47.0	47.0	47.0	47.1	47.2	47.06
3.45 to 4.15	16.5	16.5	16.5	16.5	16.5	16.5	45.5	45.5	45.6	45.6	45.7	45.58
4.15 to 4.45	16.0	16.0	16.1	16.1	16.1	16.04	43.4	43.5	43.5	43.6	43.6	43.52
4.45 to 5.15	18.5	18.5	18.6	18.6	18.6	18.50	44.4	44.5	44.6	44.7	44.8	44.60
	17.9	17.9	17.9	17.8	17.8	17.86	42.0	42.0	42.0	41.9	41.9	41.96
Proportional tank	17.9	18.0	18.0	18.1	18.1	18.02	46.6	46.7	47.0	47.0	47.0	46.86
	18.1	18.1	18.1	18.0	18.0	18.06	46.6	46.6	46.6	46.6	46.6	46.6

SHEET A 5.
Test No. 35.Date: May 28, 1909.
Observer: Wynne.

Time of taking sample.	Meter reading (cubic feet).			Weight of water (grams).	Condensed water (grams).	Gas temperature (°F.).	Barometer (inches).	Corrected gas (cubic feet).	B. t. u. per cubic foot.	
	Start.	Close.	Used.						Gross.	Net.
<i>p. m.</i>										
12.45 to 1.15	53.00	53.25	0.25	1,523	5.3	75	29.3	0.234	670	616
1.15 to 1.45	54.60	54.85	.25	1,646	6.8	75	29.3	.234	775	706
1.45 to 2.15	56.00	56.25	.25	1,555	7.0	75	29.3	.234	685	624
2.15 to 2.45	59.40	59.65	.25	1,373	6.5	75	29.3	.234	676	610
2.45 to 3.15	61.90	62.15	.25	1,200	7.0	75	29.3	.234	626	555
3.15 to 3.45	62.40	62.65	.25	1,190	7.0	76	29.3	.234	625	554
3.45 to 4.15	64.70	64.95	.25	1,177	5.8	76	29.3	.234	582	524
4.15 to 4.45	66.90	67.15	.25	1,100	6.0	76	29.3	.234	513	452
4.45 to 5.15	69.50	69.75	.25	1,025	5.0	77	29.3	.233	455	404
	73.00	73.25	.25	975	5.0	78	29.3	.232	402	351
Proportional tank	74.4	74.65	.25	1,300	7.0	79	29.3	.232	641	599
	75.0	75.25	.25	1,360	7.0	80	29.3	.231	666	594
Calculated average									654	582

SHEET A 6a.
Test No. 35.Date: May 28, 1909.
Observer: W. T. Alliger.

PHOTOMETER.

Time of taking sample.	Time burning (minutes).			Meter reading.			Gas per hour measured (cubic feet).	Gas temperature (°F.).	Gas per hour corrected (cubic feet).	Tabular No.	Candle-power.
	Start.	Close.	Length.	Start.	Close.	Used.					
<i>p. m.</i>											
12.40 to 1.10	0	5	5	45.0	12.7	17.7	3.54	80	3.26	7426	22.7
1.10 to 1.40	0	5	5	2.5	22.6	20.1	4.02	82	3.69	7791	21.1
1.40 to 2.10	0	5	5	2.0	24.9	22.9	4.58	82	4.21	7791	18.5
2.10 to 2.40	0	5	5	10.0	34.9	24.9	4.98	82	4.58	8162	17.85
2.40 to 3.10	0	5	5	25.0	52.5	27.5	5.50	82	5.05	7914	15.7
3.10 to 3.40	0	5	5	.0	28.0	28.0	5.60	82	5.15	7669	14.9
3.40 to 4.10	0	5	5	23.0	51.4	28.4	5.68	83	5.20	5795	11.1
4.10 to 4.40	0	5	5	40.0	20.7	30.7	6.14	83	5.61	9.62
4.40 to 5.10	0	5	5	40.0	19.6	29.6	5.92	84	5.40	8.12
Proportional	0	5	5	35.0	11.1	26.1	5.22	85	4.74	8037	16.9
	0	5	5	15.0	41.1	26.1	5.22	85	4.74	8067	16.9

Sheets for recording tests—Continued.

SHEET A 6b.
Test No. 35.Date: May 28, 1909.
Observer: W. T. Alliger.

PHOTOMETER—Continued.

[Average barometer, 29.3.]

Time of taking sample.	Photometer readings (mm.).										Average.
	1.	2.	3.	4.	5.	6.	7.	8.	9.	10.	
<i>p. m.</i>											
12.40 to 1.10	492	490	485	490	490	489	488	490	489	492	489
1.10 to 1.40	484	487	484	486	484	478	470	477	478	472	480
1.40 to 2.10	489	485	483	483	480	480	477	473	473	474	480
2.10 to 2.40	475	473	470	470	470	470	472	470	469	469	471
2.40 to 3.10	481	483	481	474	473	482	478	472	476	473	477
3.10 to 3.40	490	486	486	486	486	484	475	480	478	477	483
3.40 to 4.10	539	541	538	538	539	537	540	541	545	542	540
4.10 to 4.40	662	663	660	656	652	655	649	642	665	660	656
4.40 to 5.10	1,116	1,117	1,123	1,135	1,128	1,141	1,163	1,154	1,164	1,164	1,140
Proportional.....	{ 482 471	{ 478 472	{ 480 471	{ 481 468	{ 477 469	{ 469 476	{ 468 477	{ 470 478	{ 468 476	{ 469 478	{ 474 474

SHEET A 7a.
Test No. 35.Date: May 28, 1909.
Observer: W. A. Dunkley.

GAS ANALYSIS.

Time of taking sample.	Volume of gas (c. c.).	CO.		CH ₄ .		O ₂ .		CO.	
		After KOH (c. c.).	Per cent.	After Br. (c. c.).	Per cent.	After P. (c. c.).	Per cent.	After Cu ₂ Cl ₂ (c. c.).	Per cent.
<i>p. m.</i>									
12.45 to 1.15	100.0	98.8	3.2	88.4	8.4	87.8	0.6	79.1	8.7
1.15 to 1.45	100.0	98.3	3.7	88.7	7.6	88.0	.7	80.1	7.9
1.45 to 2.15	99.5	98.4	3.1	90.6	5.8	90.0	.6	81.0	9.0
2.15 to 2.45	100.0	98.8	3.2	92.0	4.8	91.2	.8	82.8	8.4
2.45 to 3.15	100.0	97.4	2.6	94.6	2.8	93.8	.8	85.4	8.4
3.15 to 3.45	99.7	97.6	2.1	95.7	1.9	94.6	1.1	87.0	7.6
3.45 to 4.15	100.0	98.4	1.6	97.5	.9	96.7	.8	89.2	7.5
4.15 to 4.45	99.6	98.4	1.2	98.4	.0	98.0	.4	91.5	6.5
4.45 to 5.15	100.0	99.0	1.0	98.8	.2	96.5	2.3	91.6	4.9
Proportional.....	99.4	96.2	3.2	92.0	4.2	90.8	1.2	83.0	7.8

SHEET A 7b.
Test No. 35.Date: May 28, 1909.
Observer: W. A. Dunkley.

Time of taking sample.	After CuCl (c. c.).	Volume for explosion (c. c.).	Air to (c. c.).	Volume after explosion (c. c.).	Contraction (c. c.).	Volume after absorption (c. c.).	Absorption (c. c.).	CH ₄ (per cent.).	H ₂ (per cent.).	N ₂ (per cent.).
<i>p. m.</i>										
12.45 to 1.15	79.1	8.0	99.3	85.4	13.9	79.9	5.5	54.4	19.1
		8.0	99.6	85.6	14.0	80.2	5.4	53.4	21.2	5.1
1.15 to 1.45	80.1	8.0	99.0	85.8	13.2	81.2	4.6	46.1	26.7
		8.0	99.4	86.1	13.3	81.5	4.6	46.1	27.3	7.0
1.45 to 2.15	81.0	8.0	99.5	86.0	13.5	82.0	4.0	40.5	37.1
		8.0	99.4	85.9	13.5	81.9	4.0	40.5	37.1	3.9
2.15 to 2.45	82.8	8.0	98.8	85.8	13.0	82.1	3.7	38.3	38.6
		8.0	97.9	85.1	12.8	81.6	3.5	36.2	40.0	6.3
2.45 to 3.15	85.4	9.0	99.6	84.6	15.0	81.0	3.6	34.2	49.3
		9.0	99.3	84.5	14.8	80.8	3.7	35.1	46.8	2.7
3.15 to 3.45	87.0	9.0	99.1	85.1	14.0	82.0	3.1	30.1	50.2
		9.0	98.8	84.9	13.9	81.7	3.2	30.9	48.4	7.5
3.45 to 4.15	89.2	9.0	99.1	85.4	13.7	83.1	2.3	22.8	60.1
		9.0	99.0	85.2	13.8	82.8	2.4	23.8	59.5	6.1
4.15 to 4.45	91.5	10.0	98.7	83.6	15.1	82.0	1.6	14.7	73.0
		10.0	99.3	84.1	15.2	82.4	1.7	15.6	72.4	4.1
4.45 to 5.15	91.6	10.0	99.3	86.6	12.7	85.9	.7	6.4	69.0
		10.0	99.0	86.4	12.6	85.7	.7	6.4	68.5	16.5
Proportional.....	{ 83.0	{ 8.0 8.0	{ 99.0 99.2	{ 86.2 86.6	{ 12.8 12.6	{ 83.0 83.4	{ 3.2 3.2	{ 33.1 33.1	{ 44.3 42.7	{ 7.0

Sheets for recording tests—Continued.

SHEET A 8a.
Test No. 35.

Date: May 28, 1909.

Observers: W. A. Dunkley, D. F. Smith.

TAB.

Time of taking sample.	Source.	Weight.				Volume.			
		Gross (gms.).	Tare (gms.).	Net (gms.).	Net (lbs.).	Jar.		Tar.	
						Total (c. c.).	Excess (c. c.).	c. c.	Gals.
12.40 to 1.10 <i>p. m.</i>	Condenser	1,421	708	713	1.567	1,720	1,100	620	0.164
	do	1,580	621	959	2.110	1,800	970	830	.220
	do	1,370	573	797	1.752	1,630	1,200	630	.166
	Separator Inlet.	640	594	246	.541	1,790	1,560	230	.061
1.10 to 1.40	Condenser	1,441	612	829	1.824	1,630	1,100	730	.183
	do	1,317	626	691	1.520	1,660	1,200	460	.122
	Separator Inlet.	703	613	90	.198	1,810	1,680	130	.033
1.40 to 2.10	Condenser	746	632	114	.251	1,770	1,660	110	.029
	do	1,515	745	770	1.695	1,700	1,000	700	.185
	Separator Inlet.	802	757	45	.099	1,820	1,780	40	.010
2.10 to 2.40	Condenser	1,340	790	550	1.210	1,810	1,320	490	.130
	Separator Inlet.	674	627	47	.103	1,750	1,740	10	.006
2.40 to 3.10	Condenser	1,108	794	314	.691	1,800	1,540	260	.069
	Separator Inlet.	417	385	32	.070	930	920	10	.003
3.10 to 3.40	Condenser	562	429	133	.292	910	800	110	.029
	Separator Inlet.	453	425	28	.062	910	910	0	.000
3.40 to 4.10	Condenser	509	428	81	.178	950	860	90	.024
	Separator Inlet.	461	448	13	.029	940	920	20	.005
4.10 to 4.40	Condenser	500	455	45	.099	930	900	30	.008
	Separator Inlet.	447	425	22	.048	920	915	5	.001
4.40 to 5.10	Condenser	398	387	11	.024	920	920	0	.000
				6,430					1.485

SHEET A 8b.
Test No. 35.

Date: May 28, 1909.

Observers: W. A. Dunkley, D. F. Smith.

Time of taking sample.	Source.	Weight.				Volume.			
		Gross (gms.).	Tare (gms.).	Net (gms.).	Net (lbs.).	Jar.		Tar.	
						Total (c. c.).	Excess (c. c.).	c. c.	Gals.
12.40 to 1.10 <i>p. m.</i>	Separator outlet	2,259	595	1,664	3.661	1,840	340	1,500	0.397
1.10 to 1.40	do	2,262	751	1,511	3.324	1,850	500	1,350	.356
1.40 to 2.10	do	1,940	606	1,332	2.930	1,800	610	1,190	.314
2.10 to 2.40	do	1,950	792	1,158	1.448	1,840	840	1,000	.265
2.40 to 3.10	do	1,822	617	1,205	2.651	1,800	900	1,000	.285
3.10 to 3.40	do	1,575	583	992	2.180	1,810	1,000	810	.214
3.40 to 4.10	do	939	435	504	1.110	930	500	430	.114
4.10 to 4.40	do	765	458	307	.675	940	700	240	.063
				8,673					1.988
				6,430					1.455
Carried from sheet No. A 8a				15,103	33.267				3.443
									5

Gallons per ton..... 17.215

$$\frac{3.443 \times 2,000}{400} = 17.2 \text{ gallons per ton of coal as charged.}$$

$$\frac{3.443 \times 2,000}{381} = 18.1 \text{ gallons per ton of dry coal.}$$

$$\frac{3.443 \times 2,000}{353} = 19.5 \text{ gallons per ton of coal ash and moisture free.}$$

Sheets for recording tests—Continued.

Sheet A 10.
Test No. 35.Date: May 28, 1909.
Observer: W. A. Dunkley.

TAR TUBES.

Time of taking sample.	Source.	Tank No.	Capacity of tank (cubic yards).	Gas made in period (cubic feet).	Weight of tar tube.		Weight of suspended tar and water.		
					Gross (grams).	Tare (grams).	Tank (grams).	Uncorrected (cubic feet).	Period.
<i>p. m.</i>									
12.40 to 1.05	P. and A. outlet.	8	1.026	157.1	30.098	28.569	1.523	1.485	233.5
1.08 to 1.32	do.	7	1.042	189.5	27.024	25.638	1.386	1.330	262.0
1.40 to 2.04	do.	8	1.026	207.6	27.927	26.887	1.040	1.014	210.5
2.08 to 2.29	do.	7	1.042	187.4	25.435	24.812	.623	.598	112.0
2.33 to 2.58	do.	8	1.026	197.3	21.826	21.242	.584	.570	112.7
3.03 to 3.24	do.	7	1.042	204.1	26.901	26.285	.616	.590	120.3
3.27 to 3.52	do.	8	1.026	235.5	30.926	30.363	.563	.549	129.2
3.57 to 4.19	do.	7	1.042	184.4	26.634	26.148	.496	.466	89.7
4.22 to 4.47	do.	8	1.026	205.9	25.869	25.456	.413	.403	83.0

Sheet A 10a.
Test No. 35.Date: May 24, 1909.
Observer: W. A. Dunkley.

NAPHTHALENE IN TARS.

Time of taking sample.	Tank No.	Capacity of tank (cubic feet).	Gas made in period (cubic feet).	Weight of U tube (grams).		Number of hours run.	C ₁₀ H ₈ naphthalene (milligrams).			
				Initial.	Final.		Corrected.	Tank.	Per cubic foot.	Period.
<i>p. m.</i>										
12.40 to 1.05	8	1.026	157.1	7.1087	7.1205	12	5.8	5.6	880
1.08 to 1.32	7	1.042	189.5	10.0009	10.0063	15	10.4	9.9	1,875
1.40 to 2.04	8	1.026	207.6	.7447	.7506	10	9.2	8.9	1,848
2.08 to 2.29	7	1.042	187.4	2.2067	2.2115	10	8.1	7.7	1,430
2.33 to 2.58	8	1.026	197.3	1.3814	1.3866	10	8.5	8.3	1,637
3.02 to 3.23	7	1.042	204.1	7.1657	7.1696	10	7.2	6.9	1,407
3.27 to 3.52	8	1.026	235.5	8.6033	8.6099	7	8.9	8.7	2,050
3.57 to 4.19	7	1.042	184.4	6.5011	6.5069	10	9.1	8.6	1,587
4.22 to 4.47	8	1.026	205.9	7.1667	7.1688	7	4.4	4.3	887

Sheet A 10b.
Test No. 35.Date: May 24, 1909.
Observer: W. A. Dunkley.

NAPHTHALENE IN GAS.

Time of taking sample.	Tank No.	Capacity of tank (cubic feet).	Gas made in period (cubic feet).	Weight of U tube (grams).		Number of hours run.	Naphthalene (milligrams).			
				Initial.	Final.		Corrected.	Tank.	Cubic feet.	Period.
<i>p. m.</i>										
12.40 to 1.05	8	1.026	157.1	7.1655	7.1677	19	8.9	8.3	1,308
1.08 to 1.32	7	1.042	189.5	2.2068	2.2170	7	12.5	11.3	2,160
1.40 to 2.04	8	1.026	207.6	6.5011	6.5087	7	9.1	8.4	1,745
2.08 to 2.29	7	1.042	187.4	10.8796	10.8817	13	6.4	5.8	1,088
2.33 to 2.58	8	1.026	197.3	10.0013	10.0067	7	9.7	8.9	1,772
3.02 to 3.23	7	1.042	204.1	10.0	2,041
3.27 to 3.52	8	1.026	235.5	1.3824	1.3882	13	10.1	9.4	2,200
3.57 to 4.19	7	1.042	184.4	9.0715	9.0753	13	8.1	7.4	1,364
4.22 to 4.47	8	1.026	205.9	8.8960	8.9044	7	10.7	9.9	2,080

COMPUTATIONS FOR ILLUMINATING-GAS TESTS.

FORM B 1.

COAL AND COKE.

I. Coal:

A. A. No. _____ } to be taken from data cards according to sample numbers on sacks.
 Locality _____ }

Screen test: Per cent screened out by $\frac{1}{8}$ -inch bar screen = $\frac{\text{weight of screenings}}{\text{weight of total coal put over screen}}$

Analyses of coal and coke to be copied from analyses of samples taken on date of test.

Weight of coal charged to be taken from data sheet.

Weight of dry coal = weight of coal \times (100 - per cent of moisture in coal).

Weight of coal, ash and moisture free = weight of coal \times (100 - per cent of moisture and ash in coal).

Length of gasifying period = time of drawing less time of charging. Express in hours and minutes and also in decimals of hours.

Example: 5:11-12:38 = 4 hrs. 33 min.

= 4.55 hrs.

Place results on Sheet C 1.

II. Coke:

Weight of dry, unquenched coke from data sheet.

Per cent of dry coke from coal as charged = $\frac{\text{weight of dry coke}}{\text{weight of coal}}$

Per cent of dry coke from dry coal = $\frac{\text{weight of dry coke}}{\text{weight of dry coal}}$

Weight of coke, ash and moisture free, to be obtained from analysis of coke by multiplying weight of dry coke by (100 - per cent ash in coke).

Per cent of coke ash and moisture free on coal ash and moisture free =

$\frac{\text{weight of coke ash and moisture free}}{\text{weight of coal ash and moisture free}}$

Weight quenched coke after draining _____ hrs. From data sheet.

Per cent moisture in quenched coke.

Screen test: Weight quenched coke screened _____ pounds.

Weight coke through screen _____ pounds.

Per cent of coke falling through $\frac{1}{8}$ -inch bar screen = coke breeze = $\frac{\text{weight of screenings}}{\text{weight of coke screened}}$

Weight of lump coke = weight of coke screened - breeze.

Place results on Sheet C 1.

FORM B 2.

RETORT TEMPERATURE.

Retort temperature, both inside and outside. Correct according to calibration curve of pyrometer, making allowance for temperature of cold junction. If readings are in centigrade scale, convert to Fahrenheit.

Plot interior and exterior curves in degrees Fahrenheit. See Sheet C 2.

Find average exterior temperature.

Place result on Sheet C 1.

FORM B 3.

PRESSURES—GAS-MADE.

Plot volume of gas made, cubic feet, uncorrected, by five-minute intervals. Sheet C 2.

Tabulate yield of gas, uncorrected, by half-hour intervals, expressing last half hour, if incomplete, fractionally.

Example: 9 ($\frac{1}{2}$) = 20. Sheet C 3.

Total gas made, uncorrected = final meter reading - initial meter reading + the volume of gas used in ammonia and naphthalene sampling.

Example: 2357 - 280 + 9 = 2086. Correct for error in station meter, if any, and for temperature and pressure according to standard tables, gas saturated with moisture at a temperature of 60° F. and a barometric pressure of 30 inches. Place on sheet C 3.

Example: Temperature of gas at outlet of meter = 54° F.

Barometer = 29.6 inches; correction factor = 1.001.

Factor on station meter = 1.00.

$2077 \times 1.00 \times 1.001 = 2079$.

Find correct yield of gas by half-hour intervals in same manner as total gas. Place on sheet C 3 and plot on Sheet C 4.

Yield of gas corrected per pound of coal as charged = $\frac{\text{corrected yield}}{\text{weight of coal charged}}$

Yield of gas corrected per pound of dry coal = $\frac{\text{corrected yield}}{\text{weight of dry coal}}$

Yield of gas corrected per pound of dry coal ash and moisture free = $\frac{\text{corrected yield}}{\text{weight of coal ash and moisture free}}$

FORM B 4.

GENERAL TEMPERATURES.

Average condenser temperatures; separator inlet and outlet temperatures.

Find average temperature of water in meter.

Find average temperature of gas at outlet to meter.

Apply calibration correction to averaged temperatures from thermometer calibration sheet. In case any readings are in the centigrade scale, convert to Fahrenheit.

FORM B 5.

CALORIMETER.

Check calculations on data sheet.

Correct gas volume to 60° F. and 30 inches barometer, according to tables.

B. t. u. gross = $\frac{\text{kilograms of water (av. outlet temperature - av. inlet temperature)} \times 3.968}{\text{corrected gas used}}$

B. t. u. net = B. t. u. - (.6 \times c. c. condensed water \times 3.968). Obtain the average B. t. u. by multiplying the cubic feet of gas made each half hour by the heating value of the respective samples, adding together the B. t. u.-feet thus found and dividing by the total number of feet of gas made. Place on Sheet C 3.

Plot gross and net B. t. u. according to Sheet C 4.

FORM B 6.

PHOTOMETER.

Check calculations on data sheet.

To obtain gas burned per hour proceed as follows:

Each revolution of the large hand is one-twelfth cubic foot, which, by observations of one minute, gives the rate in cubic feet per hour. The small hand passes over a dial graduated from 0 to 50. If the small hand does not pass over the zero during a test, the gas burned per hour (uncorrected) is obtained by subtracting the initial reading from the final and dividing by the time in minutes.

Example: 45-15=30.

Time, 6' 10"=6.16 min.

Consumption uncorrected = $\frac{30}{6.16}$ = 4.87.

If the small hand passes the zero during a test, subtract the initial reading from 50 and add it to the final.

Example: Initial 40, final 20. 50-40+20=30.

Correct consumption to 60° F. and 30 inches barometer, from temperature taken at photometer meter. Place on Sheet A 6a.

Obtain tabular number from table corresponding to average photometric reading. (This for Hefner lamp and 2,500-mm. bar.) Place on Sheet A 6a.

Candlepower = $\frac{\text{tabular number}}{\text{corrected consumption}}$

Place candlepower on Sheet C 3 and plot according to Sheet C 4.

Obtain the calculated average candlepower by multiplying the candlepower for each half hour by the cubic feet of gas made in the half hour, adding together these figures for candle-feet for all the half hours and dividing by the total make of gas. Place on Sheet C 3.

FORM B 7.

GAS ANALYSIS.

Check calculations.

CO₂, C₂H₄, O, and CO= percentages read directly when 100 c. c. sample is taken.

CH₄ = $\frac{\text{vol. after CO absorption} \times (\text{vol. after explosion} - \text{vol. after absorption of CO}_2)}{\text{vol. taken for explosion}}$

This volume CH₄ indicates percentage if 100 c. c. sample is taken.

H₂ = $\frac{\text{vol. after CO absorption} \left[\frac{3}{2} \left\{ \text{vol. after explosion} - 2 (\text{vol. after explosion} - \text{vol. after absorption}) \right\} \right]}{\text{vol. taken for explosion}}$

This gives percentage if 100 c. c. sample is taken.

Nitrogen is taken by difference between the sum of the O₂, CO₂, CO, C₂H₄, CH₄, and H₂ and 100. It should always be at least four times the oxygen, but should not be much more than 3 per cent above this figure. In case the nitrogen is excessively high or low scrutinize all the figures very carefully. Place results on Sheet C 5. Plot according to Sheet C 6.

Obtain the calculated average gas analysis by multiplying the percentage of each constituent in a half-hour sample by the cubic feet of gas made in the half hour, adding together these figures for each constituent for all half hours and dividing by the total cubic feet of gas made. Place on Sheet C 5.

FORM B 8.

TAR AND LIQUOR.

Tar from condensers. Take weight directly from Sheet A 8a for half-hour periods. The tar marked "Separator inlet" is to be counted as from condensers. Place on Sheet C 7a.

Liquor from condensers. Take weight directly from Sheet A 9a for half-hour periods. Place on Sheet C 7a.

Tar from separator outlet. Take weight directly from Sheet A 8b for half-hour intervals. Place on Sheet C 7a.

Tar in gas at outlet of separator.

The weights of suspended tar and water per cubic foot of gas, uncorrected, are obtained from Sheet A 10 and placed on Sheet C 7b, column 2. Column 1 of Sheet C 7a is obtained from column 1 of Sheet A 10; the time of emptying each aspirator tank is taken and to this time is added half of the few minutes' interval required to change tanks, it being assumed that in this interval the yield of tar does not vary materially. The weight of nonvolatile tar is the difference between the weight of the tar tube after the volatile constituents of the tar have been removed and its initial weight. Place in column 3 of Sheet C 7b. The weight of tar as sampled is calculated from this figure by multiplying by a factor to be determined experimentally for each test. The gas made in the period is not to be corrected to standard conditions, since the gas in the aspirator tank was not thus corrected. The weight of tar per period is obtained by multiplying the tar as sampled by the gas made in the period. These figures are then, for ease of comparison with other tests, calculated to even half-hour periods, a sufficient part of the second sample being added to the first to make the total equivalent to an exact half-hour run.

Example.	Period.	Weight of tar.
	28 min.	50 grams.
	31 min.	60 grams.

$$\text{Weight of tar in first half hour} = 50 + \frac{2}{31} \times 60.$$

Place in last column of sheet C 7b and also in third column of Sheet C 7c.

Total weight of tar in grams—sum of tar from condenser, tar from separator, and tar in gas passing separator.

Total tar in cubic centimeters = $\frac{\text{weight in grams}}{\text{specific gravity of average samples}}$

The specific gravity of average sample will be found on data sheet.

Gallons of tar per charge = $\frac{\text{c. c. tar}}{3,786}$

Total gallons of tar per ton of coal as charged = $\frac{\text{gallons per charge} \times 2,000}{\text{weight of charge}}$

Total gallons of tar per ton of dry coal = $\frac{\text{gallons per charge} \times 2,000}{\text{weight of dry coal charged}}$

Total gallons of tar per ton of coal ash and moisture free = $\frac{\text{gallons per charge} \times 2,000}{\text{weight of coal ash and moisture free charged}}$

Percentage of tar removed by separator by half-hour intervals = $\frac{\text{weight of tar from separator}}{\text{total tar for interval}}$

Plot total tar and liquor by half-hour intervals according to Sheet C 8.

FORM B 9.

AMMONIA.

NH₃ in liquor:

Weight of liquor for each period from data sheet.

Weight of sample = gross weight—tare weight.

Calculation: c. c. H₂SO₄ used \times factor = ——— normal.

c. c. NaOH used \times factor = ——— normal.

Subtract and difference = c. c. normal NH₃.

c. c. normal NH₃ \times .0170 = weight of NH₃ in grams.

Percentage of NH₃ in sample = $\frac{\text{weight of NH}_3}{\text{weight of sample for analysis}}$

Weight of NH₃ in liquor for period = weight of liquor \times percentage of NH₃.

Calculate percentage of NH₃ from condenser = sum of half-hour periods in grams.

NH₃ in gas at inlet of separator.

Calculation of analysis, as above, gives weight of NH₃ per tank.

NH₃ per cubic foot uncorrected = $\frac{\text{weight of NH}_3}{\text{capacity of tank}}$

It is assumed that the temperature and pressure of gas in tank are the same as in the station meter. NH₃ for intervals of emptying tank = NH₃ per cubic foot uncorrected \times uncorrected make of gas for same interval.

NH₃ for even half-hour intervals to be calculated in same manner as tar at outlet of separator. See Sheet B 8a.

Total NH₃ at inlet to separator = sum of intervals in grams.

Total NH_3 by half-hour intervals = sum of corresponding NH_3 in liquor and in gas from charge.

Total NH_3 = sum of NH_3 in liquor + NH_3 in gas.

Pounds of NH_3 per charge = $\frac{\text{weight in grams}}{453.6}$

Pounds of NH_3 per 2,000 pounds of coal as charged = $\frac{\text{pounds per charge} \times 2,000}{\text{weight of charge}}$

Pounds of NH_3 per 2,000 pounds of dry coal = $\frac{\text{pounds per charge} \times 2,000}{\text{weight of dry coal charged}}$

Pounds of NH_3 per 2,000 pounds of coal, ash and moisture free = $\frac{\text{pounds per charge} \times 2,000}{\text{weight of coal substance charged}}$

Plat total NH_3 by half-hour intervals according to Sheet C 10.

FORM B 10.

NAPHTHALENE.

Naphthalene:

Weight of naphthalene in U tubes = final weight - initial weight + correction of milligrams for each hour's run.

Naphthalene in gas (Sheet C 11a):

Weights in column 8 of Sheet A 10a indicate results per tank.

Weight of C_{10}H_8 per interval = weight per cubic foot \times gas made in interval (uncorrected).

Calculate results to even half-hour periods as in Sheet B 8. Place on Sheet C 11a.

Calculate naphthalene per cubic foot of gas corrected by dividing the naphthalene per half hour (column 6 of Sheet C 11a) by the corrected gas made per half hour (column 2 of Sheet C 3). Place on Sheet C 4d.

Tabulate saturating temperature for C_{10}H_8 in gas at outlet of separator noting whether description of corresponding tar tubes shows crystalline naphthalene. Place on Sheet C 11a.

Naphthalene in tar collected by tar tubes:

Correct the milligrams of C_{10}H_8 per period as given in last column of Sheet A 10a to even half hours.

Put on Sheet C 11a, column 3.

Copy weight of tar from column 3 of Sheet C 7c to column 2 of Sheet C 11a.

Obtain percentage of naphthalene in tar by dividing column 3 by column 1.

FORM B 11.

SULPHUR.

H_2S at outlet of scrubber:

Correct meter reading to 60° F. and 30 inches barometer according to U. G. I. tables.

Weight of cadmium sulphide $\times .235$ = weight of H_2S in grams per volume of gas shown by meter.

Weight of H_2S in grams per cubic feet of gas = $\frac{\text{weight of } \text{H}_2\text{S}}{\text{volume of gas}}$

Grains of H_2S per 100 cubic feet of gas = weight of $\text{H}_2\text{S} \times 100 \times 15.432$.

$\frac{\text{Grains } \text{H}_2\text{S}}{1.576}$ = volume of H_2S in c. c.

$\frac{\text{Volume of } \text{H}_2\text{S} \text{ in c. c.}}{28,322}$ = cubic feet of H_2S .

cubic feet of H_2S

Cubic feet of gas corrected to 29.92 inches and 32° F. = percentage of H_2S by volume.

SPECIMEN COMPUTATION SHEETS.

SHEET B 1.

COAL.

Test No. 35.

Coal A. A. No. 12; Harrisburg, Saline County, Ill.

Screenings.....27.9 per cent.

ANALYSIS OF COAL.

(See Sheet C 1.)

Weight charged	pounds..	400
Weight dry coal	do.....	381
Weight ash and moisture free	do.....	358
Length of gasifying period.....	4 hrs. 40 min. 4.67 hrs.	

SHEET B 1a.

COKE.

Test No. 35.

Weight dry, unquenched.....	pounds..	249
Per cent dry, unquenched from coal as charged.....		62.3
Per cent dry coke from dry coal.....		65.4
Weight, coke, ash and moisture free	pounds.....	
Per cent coke substance on coal substance.....		61.8
Weight quenched coke after draining 63 hours.....	pounds..	343
Per cent moisture in quenched coke.....		27.4
Per cent screenings.....		6.7

Specimen computation sheets—Continued.

SHEET B 3.

Test No. 35.

GAS YIELD BY HALF-HOUR INTERVALS.

Half-hour period.	Yield, uncorrected (cubic feet).	Yield, corrected (cubic feet).	Half-hour period.	Yield, uncorrected (cubic feet).	Yield, corrected (cubic feet).
1.....	185	178	7.....	232	223
2.....	195	187	8.....	176	169
3.....	210	202	9.....	91	87
4.....	228	219	9½.....	7	6
5.....	244	234			
6.....	246	236		1,814	1,741

Gas made, total uncorrected	cubic feet.	1,814
Samples for $C_{10}H_8$		9
Samples for NH_3		9
Samples for H_2S		1
		1,833

Average barometric pressure	inches.	29.3
Average temperature	"F.	70
Factor for conversion95
Corresponding yield $= 1,833 \times .95$	cubic feet.	1,741
Corresponding yield per pound coal as charged	do.	4.35
Corresponding yield per pound dry coal	do.	4.57
Corresponding yield per pound dry coal ash and moisture free	do.	4.93

SHEET B 5.

Test No. 35.

AVERAGE HEAT VALUE OF GAS.

Half-hour period.	Gas made (cubic feet).	Gross heat value (B. t. u.).	Gross weight (grams).	Net heat value (B. t. u.).	Net weight (grams).
1.....	178	a 850	151,200	a 780	128,800
2.....	187	775	144,900	706	132,000
3.....	202	696	140,400	624	126,100
4.....	219	676	148,000	610	123,500
5.....	234	626	146,500	555	129,800
6.....	236	582	137,300	524	123,000
7.....	223	513	114,400	452	100,700
8.....	169	455	77,000	404	68,300
9.....	87	402	35,000	351	30,500
	1,735		1,094,700		963,300
Average heat value			632		564

a Interpolating value for lost sample.

SHEET B 6.

Test No. 35.

CANDLEPOWER OF GAS.

Half-hour period.	Gas made (cubic feet).	Results averaged.		Half-hour period.	Gas made (cubic feet).	Results averaged.	
		Candle-power.	Candle feet.			Candle-power.	Candle feet.
1.....	178	22.7	4,040	7.....	223	11.1	2,470
2.....	187	21.1	3,950	8.....	169	6.2	1,050
3.....	202	18.5	3,740	9.....	87	1.2	104
4.....	219	17.8	3,900				
5.....	234	15.7	3,640		1,735	15.2	26,454
6.....	236	14.9	3,520				

Specimen computation sheets—Continued.

SHEET B 7.

Test No. 35.

GAS ANALYSES.

Half-hour period.	Gas made, cor- rected (cubic feet).	CO ₂ .		CnH _{8n} .		O ₂ .		CO.		CH ₄ .		H ₂ .		N ₂ .	
		Per cent.	Weight (grams).	Per cent.	Weight (grams).	Per cent.	Weight (grams).	Per cent.	Weight (grams).	Per cent.	Weight (grams).	Per cent.	Weight (grams).	Per cent.	Weight (grams).
1....	178	3.2	570	8.4	1,490	0.6	107	8.7	1,550	53.9	9,600	20.1	3,580	5.1	907
2....	187	3.7	691	7.6	1,420	.7	131	7.9	1,470	46.1	8,610	27.0	5,060	7.0	1,310
3....	202	3.1	625	5.8	1,170	.6	121	9.0	1,820	40.5	8,180	37.1	7,500	3.9	787
4....	219	3.2	700	4.8	1,050	.8	175	8.4	1,840	37.3	8,170	39.3	8,600	6.3	1,390
5....	234	2.6	608	2.8	655	.8	187	8.4	1,960	34.7	8,110	48.1	11,300	2.7	632
6....	236	2.1	495	1.9	449	1.1	260	7.6	1,790	30.5	7,200	49.3	11,600	7.5	1,770
7....	223	1.6	357	.9	206	.9	178	7.5	1,670	23.3	5,200	59.8	13,300	6.1	1,360
8....	199	1.2	303	.0	0	.4	68	6.6	1,100	15.1	2,550	72.7	12,300	4.1	993
9....	87	1.0	87	.2	17	2.3	200	4.9	426	6.4	556	68.7	5,970	16.5	1,440
1,735		2.5	4,336	3.7	6,457	.8	1,427	7.9	13,626	33.5	58,176	45.6	19,200	6.0	10,299

SHEET B 8.

Test No. 35.

CALCULATION OF CUBIC FEET OF GAS FOR PERIODS SHOWN ON SHEET A 10, TAKEN FROM METER READINGS OF 5-MINUTE INTERVALS.

(See Sheet A 4.)

The first period is taken from 12.40 to 1.06 p. m. To this is added $1\frac{1}{2}$ minutes of the next period, or three-tenths of the quantity of gas flowing in the 5-minute interval between 1.06 and 1.10 p. m. In the period from 1.06 $\frac{1}{2}$ to 1.36, to the readings from 1.10 to 1.35 are added $\frac{3.5}{5}$ of the former period and one-fifth of the period following.

$$12.40 \text{ to } 1.06\frac{1}{2} \text{ p. m.:}$$

$$154 + \frac{1\frac{1}{2} \times 3.1}{5} = 157.1.$$

$$1.06\frac{1}{2} \text{ to } 1.36:$$

$$\frac{3.5 \times 31}{5} + (346 - 185) + \frac{1}{5} \times 34 = 189.5.$$

$$1.36 \text{ to } 2.06:$$

$$\frac{4}{5} \times 34 + (553 - 390) + \frac{37}{5} = 207.6.$$

$$2.06 \text{ to } 2.31:$$

$$\frac{4 \times 37}{5} + (740 - 590) + \frac{1 \times 39}{5} = 187.4.$$

$$2.31 \text{ to } 3.00\frac{1}{2}:$$

$$\frac{4 \times 39}{5} + (980 - 818) + \frac{.5 \times 41}{5} = 197.3.$$

$$3.00\frac{1}{2} \text{ to } 3.25\frac{1}{2}:$$

$$\frac{4.5 \times 41}{5} + (1184 - 1021) + \frac{.5 \times 42}{5} = 204.1.$$

$$3.25\frac{1}{2} \text{ to } 3.54\frac{1}{2}:$$

$$\frac{4.5 \times 42}{5} + (1385 - 1226) + \frac{4.5 \times 43}{5} = 235.5.$$

$$3.54\frac{1}{2} \text{ to } 4.20:$$

$$\frac{.5 \times 43}{5} + (1605 - 1428) + \frac{.5 \times 31}{5} = 184.4.$$

$$4.20\frac{1}{2} \text{ to } 5.20:$$

$$\frac{4.5 \times 31}{5} + (1814 - 1636) = 205.9.$$

Specimen computation sheets—Continued.

SHEET B 8a

Test No. 35.

CALCULATION OF TAR SUSPENDED IN GAS AT OUTLET OF SEPARATOR FOR HALF-HOUR PERIODS, FROM WEIGHTS OF TAR PER PERIOD AS SAMPLED.

(See Sheet C 7c.)

The duration of the first period was 26.5 minutes. To calculate for 30-minute period, 3.5 minutes of the next period are added to this, i. e., in the next period 48 grams of tar were delivered, and the rate per minute for this period is $\frac{48}{29.5}$. For 3 minutes it will be $3.5 \times \frac{48}{29.5}$.

For the half-hour period No. 2, 29—3.5=26.0 minutes are taken plus 4 minutes of the period following.

$$(1) 44.7 + \frac{3.5 \times 48}{29.5} = 50.4.$$

$$(2) \frac{26.0 \times 48}{29.5} + \frac{4 \times 65.4}{30} = 51.0.$$

$$(3) \frac{26 \times 65.4}{30} + \frac{4 \times 46.7}{27} = 63.5.$$

$$(4) \frac{23 \times 46.7}{27} + \frac{7 \times 47.3}{29.5} = 51.0.$$

$$(5) \frac{22.5 \times 47.3}{29.5} + \frac{7.5 \times 46.5}{25} = 50.0.$$

$$(6) \frac{17.5 \times 46.5}{25} + \frac{12.5 \times 45}{29} = 52.0.$$

$$(7) \frac{16.5 \times 45}{29} + \frac{13.5 \times 19.9}{26} = 52.5.$$

$$(8) \frac{12.5 \times 19.9}{26} + \frac{17.5 \times 20.5}{59.5} = 15.6.$$

$$(9) \frac{30 \times 20.5}{59.5} = 10.3 = 14.4$$

$$\frac{12 \times 20.5}{59.5} = 4.1.$$

SHEET B 9.

AMMONIA.

Test No. 35.

Quantity of gas taken for determining ammonia.

Time of sampling.	Meter readings (cubic feet).		Time of sampling.	Meter readings (cubic feet).	
	Cor- rected.	Uncor- rected.		Cor- rected.	Uncor- rected.
12.40 to 1.04 p. m.	148	149	3.35 to 4.03 p. m.	1,488	222
1.04 to 1.36 p. m.	353	206	4.03 to 4.42 p. m.	1,726	239
1.36 to 2.07 p. m.	568	216	4.42 to 5.12 p. m.	1,809	84
2.07 to 2.33 p. m.	764	197			
2.33 to 3.04 p. m.	1,013	250		1,818	1,818
3.04 to 3.35 p. m.	1,267	255			

Aspirator tanks, 9.

Calculation of ammonia in gas for half-hour intervals.

(See Sheet C 9a.)

Time of sampling.	Weight of NH ₃ .	
	Weight for period (grams).	Corrected weight (grams).
12.40 to 1.04 p. m.	32.6	24—24 = 32.6 } 40.2
1.04 to 1.36 p. m.	40.8	32—6 = 7.6 } 40.0
1.36 to 2.07 p. m.	53.0	26—32.2 = 4.8 } 52.7
2.07 to 2.33 p. m.	56.5	31—4 = 6.8 } 54.2
2.33 to 3.05 p. m.	19.1	27—46.2 = 3.8 } 30.6
3.05 to 3.35 p. m.	94.5	26—3 = 6.5 } 87.9
3.35 to 4.04 p. m.	52.5	23—50.0 = 2.7 } 55.0
4.04 to 4.42 p. m.	73.5	32—7 = 4.2 } 58.1
4.42 to 5.12 p. m.	21.6	25—14.9 = 10.1 } 25.4

Specimen computation sheets—Continued.

SHEET B 11b.

Test No. 35.

WEIGHT OF NAPHTHALENE.*Weight of naphthalene ($C_{10}H_8$) in tar for half-hour periods.*

$$\begin{aligned} \frac{.880 - 3.5 \times 1.875}{29.5} &= 1.18. \\ \frac{26 \times 1.875}{29.5} + \frac{4 \times 1.848}{30} &= 1.896. \\ \frac{26 \times 1.848}{30} + \frac{4 \times 1.430}{27} &= 1.81. \\ \frac{23 \times 1.430}{27} + \frac{7 \times 1.637}{29.5} &= 1.606. \\ \frac{22.5 \times 1.637}{29.5} + \frac{7.5 \times 1.407}{25} &= 1.713. \end{aligned}$$

$$\begin{aligned} \frac{17.5 \times 1.407}{25} + \frac{12.5 \times 2.050}{29} &= 1.875. \\ \frac{16.5 \times 2.050}{29} + \frac{13.5 \times 1.587}{26} &= 1.992. \\ \frac{12.5 \times 1.587}{26} + \frac{17.5 \times .887}{59.5} &= .996. \\ \frac{42.0 \times .887}{59.5} &= .628. \end{aligned}$$

Weight of naphthalene in ($C_{10}H_8$) in gas for half-hour intervals.

$$\begin{aligned} 1.304 + \frac{3.5 \times 2.27}{29.5} &= 1.573. \\ \frac{26 \times 2.15}{29.5} + \frac{4 \times 1.745}{30} &= 2.13. \\ \frac{26 \times 1.745}{30} + \frac{4 \times 1.088}{27} &= 1.71. \\ \frac{23 \times 1.088}{27} + \frac{7 \times 1.772}{29.5} &= 1.35. \\ \frac{22.5 \times 1.772}{29.5} + \frac{7.5 \times 2.04}{25} &= 1.96. \end{aligned}$$

$$\begin{aligned} \frac{17.5 \times 2.04}{25} + \frac{12.5 \times 2.20}{29} &= 2.38. \\ \frac{16.5 \times 2.20}{29} + \frac{13.5 \times 1.364}{26} &= 1.96. \\ \frac{12.5 \times 1.364}{26} + \frac{17.5 \times 2.08}{59.5} &= 1.26. \\ \frac{42 \times 2.08}{59.5} &= 1.47. \end{aligned}$$

SHEET B 11d.

Test No. 35.

NAPHTHALENE DATA.

Half-hour period.	Weight of naphthalene (grams).	Yield of gas corrected (cubic feet).	Weight of naphthalene per cubic foot (grams).	Temperature of naphthalene saturated ($^{\circ}$ F.).
1.....	1.57	178	8.8	60
2.....	2.13	187	11.4	64
3.....	1.71	202	8.5	60
4.....	1.35	219	6.2	54
5.....	1.96	234	8.4	60
6.....	2.38	236	10.1	63
7.....	1.96	223	8.8	60
8.....	1.26	169	7.5	59
9.....	1.47	93	15.8	70
Total.....			85.5	550
Average.....			9.5	61

TEMPERATURE AND WEIGHT OF SATURATED NAPHTHALENE VAPOR.

Temperature ($^{\circ}$ F.).	Weight (grams).
70	15.5
60	8.4
50	7.1
	4.5
	3.9

SHEETS FOR RECORDING COMPUTED DATA.

SHEET C 1.
Test No. 35.

Date: May 28, 1909.
Coal: Harrisburg, Ill.

RETORT DATA.

Coal A. A. No. 12.

Mine location: Harrisburg, Ill.

Per cent screened out by $\frac{3}{4}$ -inch bar screen: 27.9.

Weight of coal charged..... pounds.. 400

Weight of dry coal..... do... 381

Weight of coal ash and moisture free..... do... 353

Weight of dry coke..... do... 249

Weight of coke drawn, ash and moisture free..... do... 218

Per cent on coal charged..... 62.3

Per cent on dry coal..... 65.4

Per cent on coal substance..... 61.8

Length of gasifying period..... 4 hrs. 40 min. (4.67 hrs.)

Average temperature outside retort..... ° F.. 1,779

For make of gas and retort temperature, both inside and outside, see curves, Sheet C 2 (fig. 7).

Moisture in quenched coke after draining 63 hours..... per cent.. 27.4

Coke breeze (through $\frac{1}{4}$ -inch bar screen)..... do... 6.7

Lump coke..... do... 93.3

Remarks on appearance of coal: Over $\frac{1}{4}$ -inch screen; bright and clean; lumps up to 5 inches.

Remarks on appearance of coke: Dark color; easily broken.

Remarks on condition of retort: Badly covered on top with carbon; temperature low.

ANALYSIS OF COAL AND COKE.

	Coal as charged.	Dry coal.	Dry coke.
Moisture.....	4.66		
Ash.....	7.19	7.54	12.30
Volatile matter.....	34.44	36.13	2.22
Fixed carbon.....	53.71	56.33	85.48
Sulphur separately determined.....	100.00	100.00	100.00
B. t. u. per pound.....	1.96	2.06	2.08
	12,919	13,550	12,427

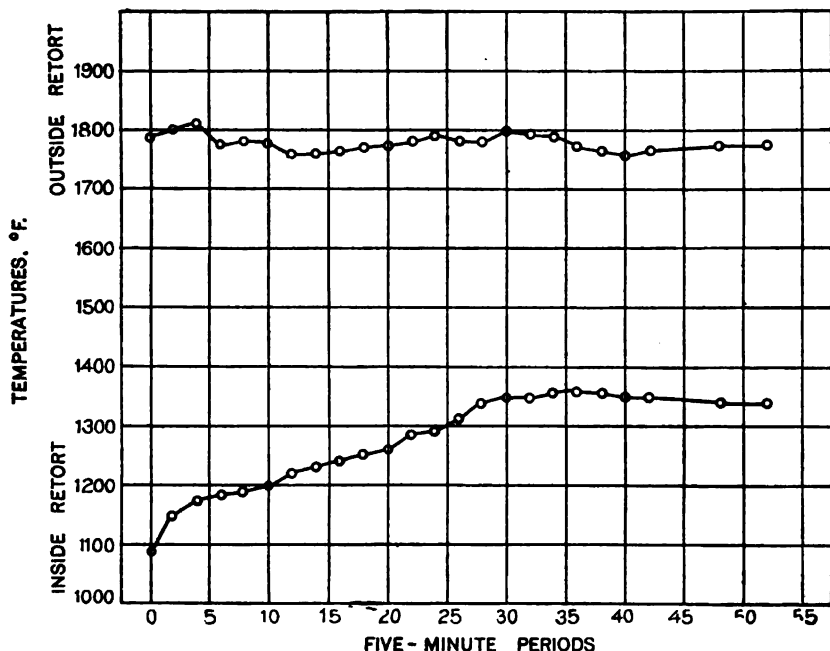


FIGURE 7.—Retort temperatures, 5-minute periods, test 35 (Sheet C 2).

Sheets for recording computed data—Continued.

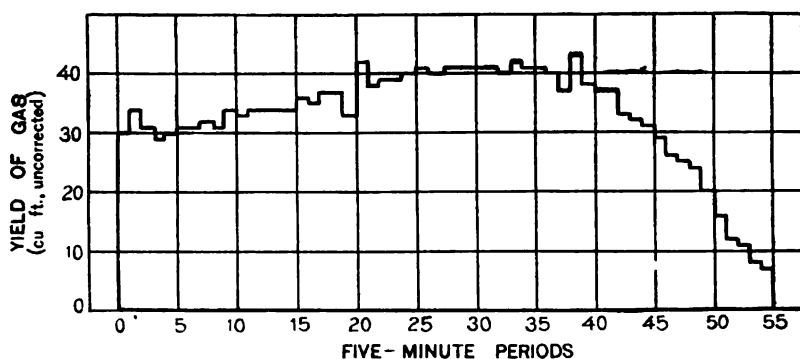


FIGURE 8.—Yield of gas, 5-minute periods, test 35 (Sheet C 2).

SHEET C 3.

Test No. 35.

Date: May 28, 1909.

Coal: Harrisburg, Ill.

GAS MADE.

Total quantity of gas made (corrected).....cubic feet.. 1,741

Gas per pound of coal as charged.....do.... 4.35

Gas per pound of dry coal.....do.... 4.57

Gas per pound of coal, ash and moisture free.....do.... 4.93

For make of gas (uncorrected) by 5-minute intervals, see curve, Sheet C 2 (fig. 8).

Half-hour period.	Gas made, corrected (cubic feet).		Candle-power.	Heating value (B. t. u.).	
	Per charge.	Per 2,000 pounds coal charged.		Gross.	Net.
1.....	178	890	22.7	^a 670	^a 616
2.....	187	935	21.1	775	706
3.....	202	1,010	18.5	695	624
4.....	219	1,095	17.8	676	610
5.....	234	1,170	15.7	626	555
6.....	236	1,180	14.9	582	524
7.....	223	1,115	11.1	513	452
8.....	169	845	6.2	455	404
9.....	87	435	1.2	402	351
9½.....	6	30			
Average.....			15.2	^b 632	^b 568
Proportional sample.....			16.9	^c 654	^d 582

^a Leak in connections; no good.^b In calculating these averages, the value for test half hours was interpolated as 850 gross and 780 net.^c Average of two determinations, 641 and 666.^d Average of two determinations, 569 and 594.

For graphic representation of above, see curves, Sheet C 4 (fig. 9).

Remarks on any of above:

Sheets for recording computed data—Continued.

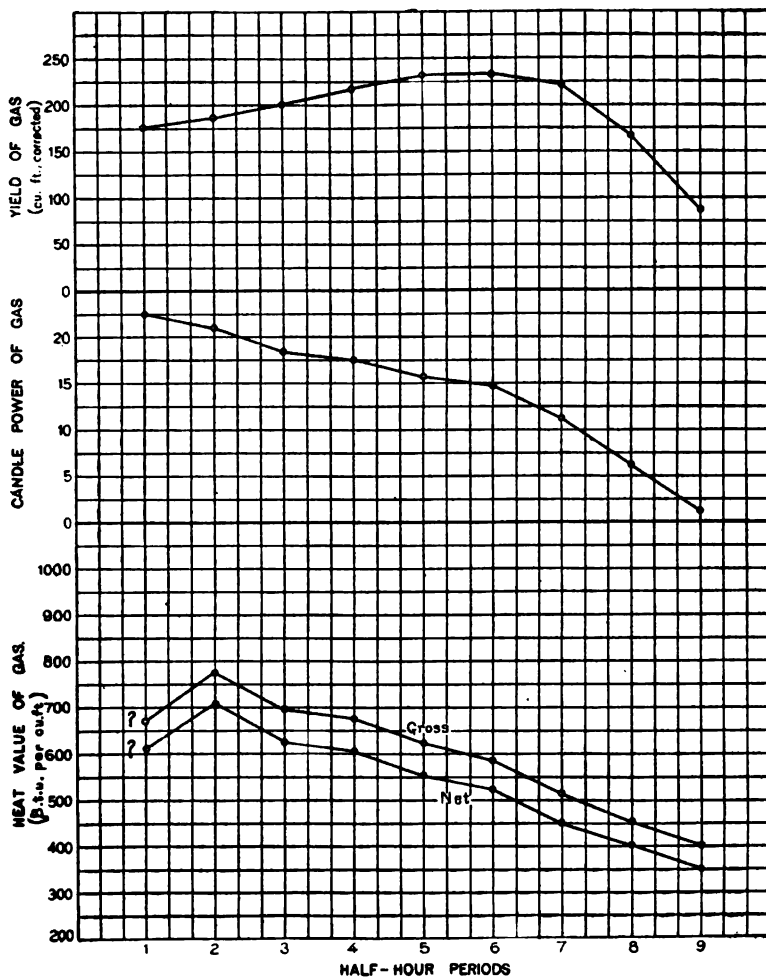


FIGURE 9.—Yield and quality of gas, half-hour periods, test 35 (Sheet C 4).

SHEET C 5.

Test No. 35.

Date: May 28, 1909.

Coal: Harrisburg, Ill.

GAS ANALYSES.

Half-hour period.	CO ₂ .	C ₂ H ₄ , etc.	O ₂ .	CO.	CH ₄ .	H ₂ .	N ₂ .
1.....	3.2	8.4	0.6	8.7	53.9	20.1	5.1
2.....	3.7	7.6	.7	7.9	46.1	27.0	7.0
3.....	3.1	5.8	.6	9.0	40.5	37.1	3.9
4.....	3.2	4.8	.8	8.4	37.3	39.3	6.3
5.....	2.6	2.8	.8	8.4	34.7	48.1	2.7
6.....	2.1	1.9	1.1	7.6	30.5	49.3	7.5
7.....	1.6	.9	.8	7.5	23.3	59.8	6.1
8.....	1.2	.0	.4	6.5	15.1	72.7	4.1
9.....	1.0	.2	2.3	4.9	6.4	68.7(?)	16.5(?)
Proportional tank.....	3.2	4.2	1.2	7.8	33.1	43.8	7.0
Computed average.....	2.5	3.7	.8	7.9	33.5	45.6	6.0

For graphic representation of above, see curves, Sheet C 6 (fig. 10).

Remarks on any of above:

Sheets for recording computed data—Continued.

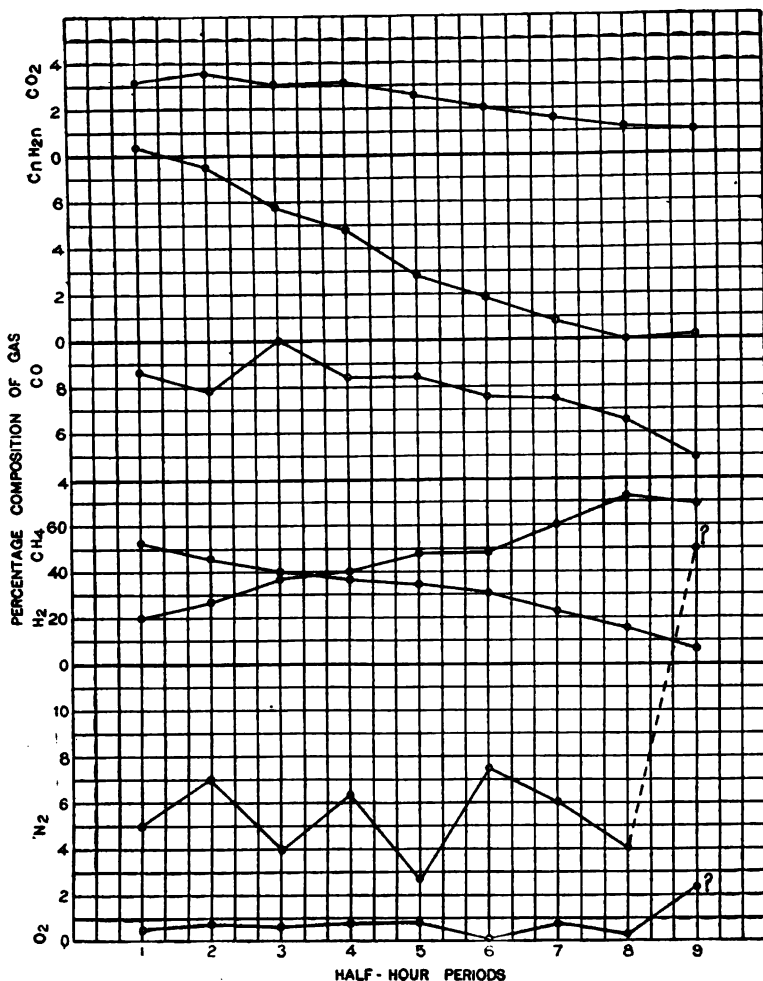


FIGURE 10.—Composition of gas, half-hour periods, test 35 (Sheet C 6).

SHEET C 7a.
Test No. 35.Date: May 23, 1909.
Coal: Harrisburg, Ill.**WEIGHT OF LIQUOR AND TAR.****WEIGHT OF AMMONIACAL LIQUOR AND TAR FROM COAL CHARGED, BY HALF-HOUR INTERVALS (GRAMS).**

Half-hour period.	Weight of liquor from condenser.	Weight of tar.			
		From condenser.	From separator.	Passing separator.	Total.
1.....	3,602	2,715	1,664	50.4	4,429.4
2.....	2,320	1,610	1,511	51.0	3,172.0
3.....	1,447	929	1,332	63.5	2,324.5
4.....	887	597	1,158	51.0	1,806.0
5.....	500	346	1,205	50.0	1,601.0
6.....	197	161	992	52.0	1,205.0
7.....	116	94	504	52.5	650.5
8.....	61	67	307	15.6	389.6
9.....		11		14.4	25.4
	9,130	6,530	8,673	400.4	15,603.4

*Sheets for recording computed data—Continued.***WEIGHT OF LIQUOR AND TAR—Continued.**

WEIGHT OF AMMONIACAL LIQUOR AND TAR PER TON OF COAL AS CHARGED, BY HALF-HOUR INTERVALS (POUNDS).

Half-hour period.	Weight of liquor from condenser.	Weight of tar.			
		From condenser.	From separator.	Passing separator.	Total.
1.....	39.70	29.85	18.30	0.55	48.70
2.....	26.57	17.71	16.62	.55	34.88
3.....	15.98	10.22	14.65	.69	25.56
4.....	9.77	6.56	7.24	.55	14.35
5.....	5.51	3.80	13.25	.55	17.60
6.....	2.17	1.67	10.90	.57	13.14
7.....	1.28	1.03	5.55	.57	7.15
8.....	.67	.73	3.37	.17	4.27
9.....		.12		.16	.28
	100.65	71.69	89.88	4.36	165.93

For graphic representation of above, see Sheet C 8 (fig. 11).

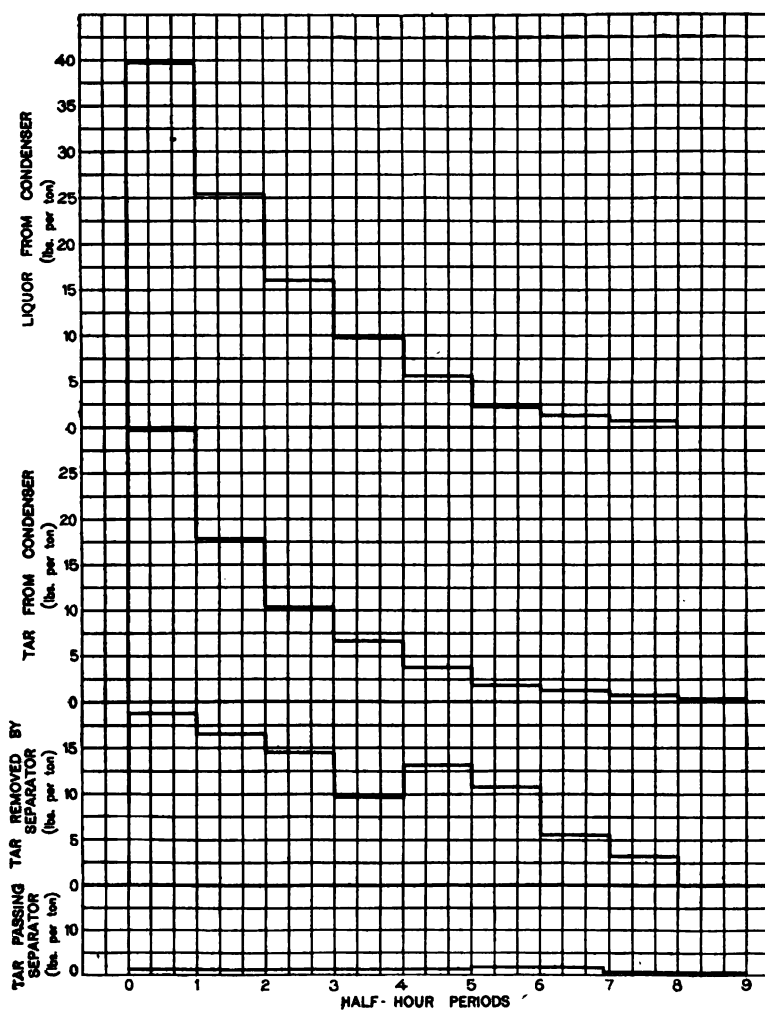


FIGURE 11.—Yield of liquor and tar, half-hour periods, test 35 (Sheet C 8).

*Sheets for recording computed data—Continued.*SHEET C 7b.
Test No. 35.Date: May 28, 1909.
Coal: Harrisburg, Ill.**TAR AT INLET AND OUTLET OF SEPARATOR.**

[All weights in grams.]

AT INLET.

[Not taken.]

AT OUTLET.

Sampling period and time out (minutes).	Weight of suspended material per cubic foot of gas, uncorrected.			Gas made in sampling period per cubic foot, uncorrected.	Weight of tar as sampled.	
	Tar and water.	Nonvolatile tar.	Tar as sampled, calculated. ^a		Per sampling period.	Per half-hour period, calculated.
26.5	1.485	0.190	0.285	157.1	44.7	50.4
29.5	1.330	.169	.253	189.5	48.0	51.0
30.0	1.014	.210	.315	207.6	65.4	63.5
27.0	.598	.166	.249	187.4	46.7	51.0
29.5	.570	.160	.240	197.3	47.3	50.0
25.0	.590	.152	.228	204.1	46.5	52.0
29.0	.549	.128	.192	235.5	45.0	52.5
26.0	.466	.072	.108	184.4	19.9	15.6
59.5	.403	.067	.100	205.9	20.5	^b 14.4

^a Tar as sampled equals nonvolatile tar multiplied by 1.5, as determined.^b 42 minutes.SHEET C 7c.
Test No. 35.Date: May 28, 1909.
Coal: Harrisburg, Ill.**ELIMINATION OF TAR BY SEPARATOR.**

Half-hour period.	Actual weight of tar removed (grams).	Calculated weight of tar—		Per cent of sum of tar removed.	Average temperature at inlet (°F.).
		At outlet (grams).	At inlet (grams).		
1	1,664	50.4	1,714.4	97.0	159
2	1,511	51.0	1,562.0	96.7	128
3	1,332	63.5	1,395.5	95.5	105
4	1,158	51.0	1,209.0	95.8	99
5	1,205	50.0	1,255.0	98.0	106
6	992	52.0	1,044.0	95.0	106
7	504	52.5	556.5	90.5	101
8	307	15.6	322.6	95.4	93
9	0 (?)	14.4	14.4	89
	8,673	400.4	9,073.4	95.2	110

Sheets for recording computed data—Continued.

SHEET C 9a.
Test No. 35.Date: May 28, 1909.
Coal: Harrisburg, Ill.

AMMONIA.

AMMONIA IN LIQUOR FROM CONDENSERS.

Length of sampling period (minutes).	NH ₃ in sample (per cent).	Weight of NH ₃ .			
		Total, in sample (grams).	Per minute, calculated (grams).	Per half-hour period, calculated.	
				Grams.	Pounds.
30	1.19	42.90	1.43	(1) 42.90	0.0945
30	.82	21.26	.71	(2) 21.26	.0468
30	1.96	28.40	.94	(3) 28.40	.0625
30	2.71	24.10	.80	(4) 24.10	.0530
30	2.40	12.00	.40	(5) 12.00	.0264
30	2.00	3.94	.13	(6) 3.94	.0087
30	2.08	2.39	.08	(7) 2.39	.0053
30	2.10	1.28	.04	(8) 1.28	.0028
				136.27	.2998

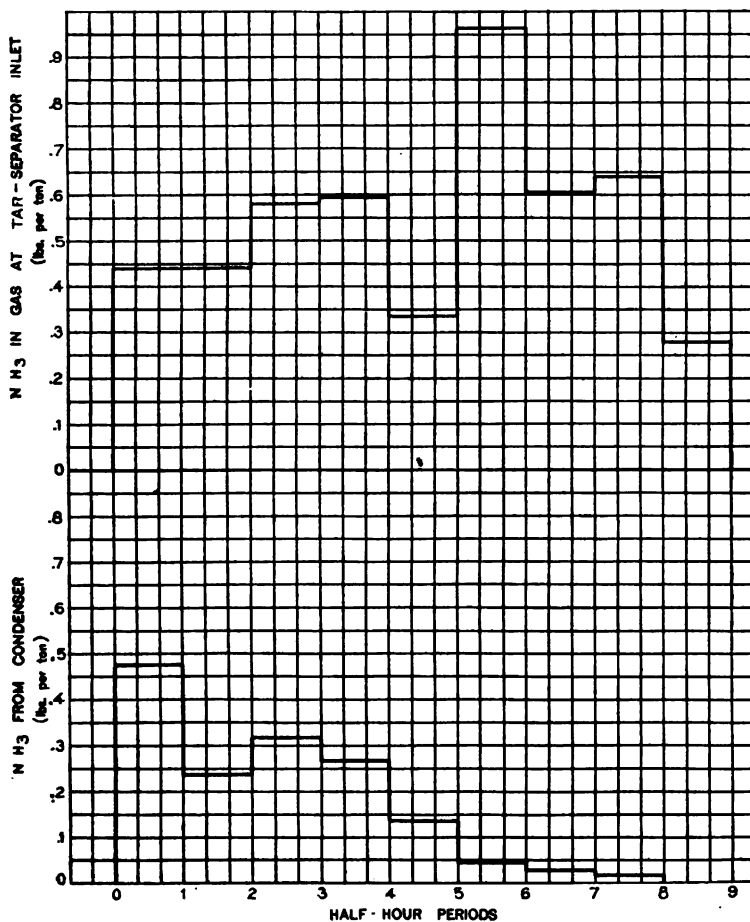


FIGURE 12.—Yield of ammonia, half-hour periods, test 35 (Sheet C 10).

Sheets for recording computed data—Continued.

AMMONIA—Continued.

AMMONIA IN GAS AT INLET TO SEPARATOR.

Sampling period and time out (minutes).	Weight of NH ₃ per cubic foot of gas (grams).	Gas made in period, uncorrected (cubic feet).	Weight of NH ₃ in sampling period (grams).	Weight of NH ₃ per half-hour period, calculated.	
				Grams.	Pounds.
24	0.219	149	32.6	(1) 40.2	0.0885
32	.198	206	40.8	(2) 40.0	.0680
31	.245	216	53.0	(3) 52.7	.1160
26	.287	197	56.5	(4) 54.2	.1190
32	.076	250	19.1	(5) 30.6	.0674
30	.370	255	94.5	(6) 87.9	.1930
29	.236	222	52.5	(7) 55.0	.1210
38	.307	239	73.5	(8) 58.1	.1280
30	.257	84	21.6	(9) 25.4	.0559
.....	444.1	444.1	.9769

AMMONIA PER CHARGE AND PER TON OF COAL, BY HALF-HOUR PERIODS.

Half-hour period.	NH ₃ per charge (pounds).		NH ₃ per ton of coal (pounds).		
	From condenser.	At separator inlet.	From condenser.	At separator inlet.	Total.
1.....	0.0945	0.0885	0.472	0.442	0.914
2.....	.0468	.0880	.234	.440	.674
3.....	.0625	.1160	.312	.580	.892
4.....	.0530	.1190	.265	.595	.860
5.....	.0264	.0674	.132	.337	.469
6.....	.0087	.1930	.043	.965	1.008
7.....	.0053	.1210	.026	.605	.631
8.....	.0028	.1280	.014	.640	.654
9.....0559280	.280
	.2998	.9769	1.500	4.88	6.38

Total NH₃ per 2,000 pounds of coal as charged.....pounds.. 6.38Total NH₃ per 2,000 pounds of dry coal.....do..... 6.70Total NH₃ per 2,000 pounds of coal, ash and mixture pee.....do..... 7.2

For graphic representation of ammonia yield, see Sheet C 10 (fig. 12).

SHEET C 11a.
Test No. 35.Date: May 24, 1909.
Coal: Harrisburg, Ill.

NAPHTHALENE AT OUTLET OF SEPARATOR.

NAPHTHALENE DISSOLVED IN SUSPENDED AIR.

[All weights in grams.]

Half-hour period.	Weight of tar from Sheet C 7c.	Per cent of naphthalene in tar.	Weight of naphthalene in tar.
1.....	50.4	2.3	1.180
2.....	51.0	3.7	1.896
3.....	63.5	2.8	1.810
4.....	51.0	3.1	1.608
5.....	50.0	3.4	1.713
6.....	52.0	3.6	1.875
7.....	52.5	3.8	1.992
8.....	15.6	6.2	.966
9 (42 minutes).....	14.4	4.4	.628
	400.4	Av. 3.7	13.668

Sheets for recording computed data—Continued.

NAPHTHALENE AT OUTLET OF SEPARATOR—Continued.

NAPHTHALENE AS VAPOR IN GAS.

Sampling period and time out (minutes).	Weight of naphthalene per cubic foot of gas, uncorrected.	Gas made in period, uncorrected (cubic feet).	Weight of naphthalene in sampling period.	Weight of naphthalene calculated to half-hour periods.
24.5	.0083	157	1.303	(1) 1.57
26.5	.0113	189	2.150	(2) 2.13
30.0	.0084	207	1.745	(3) 1.71
27.0	.0085	187	1.088	(4) 1.35
29.5	.0089	197	1.772	(5) 1.96
25.0	.0100	204	2.041	(6) 2.88
22.0	.0094	235	2.200	(7) 1.96
26.0	.0074	184	1.364	(8) 1.26
59.5	.0099	206	2.080	(9) 1.47
				15.79

SHEET C 11b.

Test No. 35.

Date: May 24, 1909.

Coal: Harrisburg, Ill.

TOTAL WEIGHT OF NAPHTHALENE.

Weight of tar from condenser.....	grams..	6,530
Weight of tar from separator.....	do....	8,673
		15,208
Per cent of naphthalene in condenser tar.....		2.5
Per cent of naphthalene in separator tar.....		
		380.00
Weight of tar passing separator as mist.....	grams..	400
Per cent of naphthalene in tar.....		3.7
		14.80
Weight of naphthalene as vapor in gas passing separator.....	grams..	15.79
Total weight of naphthalene per charge of coal.....	do....	410.59
Total weight of naphthalene per 2,000 pounds of—		
Coal as charged.....	do....	2,042.00
Dry coal.....	do....	
Coal substance.....	do....	

SHEET C 11c.

Test No. 35.

Date: May 24, 1909.

Coal: Harrisburg, Ill.

NAPHTHALENE AND SULPHUR.

NAPHTHALENE PER CUBIC FOOT OF GAS AT SEPARATOR.

(All weights in milligrams.)

Half-hour period.	Naphthalene as vapor in gas per cubic foot, corrected.		Temperature of naphthalene saturated.		Average temperature of separator.	
	Inlet.	Outlet.	Inlet.	Outlet.	Inlet.	Outlet.
1.....		8.8		60	See Sheet C 7c.	130
2.....		11.4		64		116
3.....		8.5		60		103
4.....		6.2		54		100
5.....		8.4		60		101
6.....		10.1		63		99
7.....		8.8		60		96
8.....		7.5		59		93
9.....		15.8		70		92
		9.5				105

SULPHUR.

Hydrogen sulphide at outlet of fresh-water scrubber:

Per cubic foot.....	milligrams..	900
Per 100 cubic feet.....	grains..	1,300

Hydrogen sulphide at outlet of iron-oxide purifier:

Test with lead acetate paper.....	Negative.
-----------------------------------	-----------

Total sulphur in purified gas determined on proportional sample:

Per cubic foot.....	milligrams..	Not taken.
Per 100 cubic feet.....	grains..	Not taken.

PUBLICATIONS ON FUEL TESTING.

The following publications, except those to which a price is affixed, can be obtained free by applying to the Director of the Bureau of Mines, Washington, D. C. The priced publications can be purchased from the Superintendent of Documents, Government Printing Office, Washington, D. C.

PUBLICATIONS OF THE BUREAU OF MINES.

BULLETIN 1. The volatile matter of coal, by H. C. Porter and F. K. Ovit. 1910. 56 pp., 1 pl.

BULLETIN 2. North Dakota lignite as a fuel for power-plant boilers, by D. T. Randall and Henry Kreisinger. 1910. 42 pp., 1 pl.

BULLETIN 3. The coke industry of the United States as related to the foundry, by Richard Moldenke. 1910. 32 pp.

BULLETIN 4. Features of producer-gas power-plant development in Europe, by R. H. Fernald. 1910. 27 pp., 4 pls.

BULLETIN 5. Washing and coking tests of coal at Denver, Colo., by A. W. Belden, G. R. Delamater, J. W. Groves, and K. M. Way. 1910. 62 pp.

TECHNICAL PAPER 1. The sampling of coal in the mine, by J. A. Holmes. 1911. 16 pp.

TECHNICAL PAPER 2. The escape of gas from coal, by H. C. Porter and F. K. Ovit. 14 pp.

TECHNICAL PAPER 3. Specifications for the purchase of fuel oil for the Government, with directions for sampling oil and natural gas, by I. C. Allen. 1911. 13 pp.

PUBLICATIONS OF THE UNITED STATES GEOLOGICAL SURVEY.

PROFESSIONAL PAPER 48. Report on the operations of the coal-testing plant of the United States Geological Survey at the Louisiana Purchase Exposition, St. Louis, Mo., 1904; E. W. Parker, J. A. Holmes, M. R. Campbell, committee in charge. 1906. In three parts. 1492 pp., 13 pls. \$1.50.

BULLETIN 261. Preliminary report on the operations of the coal-testing plant of the United States Geological Survey at the Louisiana Purchase Exposition, St. Louis, Mo., 1904; E. W. Parker, J. A. Holmes, M. R. Campbell, committee in charge. 1905. 172 pp. 10 cents.

BULLETIN 290. Preliminary report on the operations of the fuel-testing plant of the United States Geological Survey at St. Louis, Mo., 1905, by J. A. Holmes. 1906. 240 pp. 20 cents.

BULLETIN 325. A study of four hundred steaming tests made at the fuel-testing plant, St. Louis, Mo., 1904, 1905, and 1906, by L. P. Breckenridge. 1907. 196 pp. 20 cents.

BULLETIN 332. Report of the United States fuel-testing plant at St. Louis, Mo., January 1, 1906, to June 30, 1907; J. A. Holmes, in charge. 1908. 299 pp. 25 cents.

BULLETIN 334. The burning of coal without smoke in boiler plants; a preliminary report, by D. T. Randall. 1908. 26 pp. 5 cents.

BULLETIN 336. Washing and coking tests of coal and cupola tests of coke, by Richard Moldenke, A. W. Belden, and G. R. Delamater. 1908. 76 pp. 10 cents.

BULLETIN 343. Binders for coal briquets, by J. E. Mills. 1908. 56 pp.

BULLETIN 362. Mine sampling and chemical analyses of coals tested at the United States fuel-testing plant, Norfolk, Va., in 1907, by J. S. Burrows. 1908. 23 pp. 5 cents.

BULLETIN 363. Comparative tests of run-of-mine and briquetted coal on locomotives, including torpedo-boat tests and some foreign specifications for briquetted fuel, by W. F. M. Goss. 1908. 57 pp., 4 pls.

BULLETIN 366. Tests of coal and briquets as fuel for house-heating boilers, by D. T. Randall. 1908. 44 pp., 3 pls.

BULLETIN 367. The significance of drafts in steam-boiler practice, by W. T. Ray and Henry Kreisinger. 1909. 61 pp.

BULLETIN 368. Washing and coking tests of coal at Denver, Colo., by A. W. Belden, G. R. Delamater, and J. W. Groves. 1909. 54 pp., 2 pls. 10 cents.

BULLETIN 373. The smokeless combustion of coal in boiler plants, by D. T. Randall and H. W. Weeks. 1909. 188 pp. 20 cents.

BULLETIN 382. The effect of oxygen in coal, by David White. 1909. 74 pp., 3 pls.

BULLETIN 385. Briquetting tests at the United States fuel-testing plant, Norfolk, Va., 1907-8, by C. L. Wright. 1909. 41 pp., 9 pls.

BULLETIN 392. Commercial deductions from comparisons of gasoline and alcohol tests on internal-combustion engines, by R. M. Strong. 1909. 38 pp.

BULLETIN 402. The utilization of fuel in locomotive practice, by W. F. M. Goss. 1909. 28 pp.

BULLETIN 403. Comparative tests of run-of-mine and briquetted coals on the torpedo-boat *Biddle*, by W. T. Ray and Henry Kreisinger. 1909. 49 pp.

BULLETIN 412. Tests of run-of-mine and briquetted coal in a locomotive boiler, by Walter T. Ray and Henry Kreisinger. 1909. 32 pp.

BULLETIN 416. Recent development of the producer-gas power plant in the United States, by R. H. Fernald. 1909. 82 pp., 2 pls. 15 cents.

BULLETIN 428. The purchase of coal by the Government under specifications, by G. S. Pope. 1910. 80 pp.



Bulletin 7

DEPARTMENT OF THE INTERIOR

BUREAU OF MINES

JOSEPH A. HOLMES, DIRECTOR

ESSENTIAL FACTORS
IN THE
FORMATION OF PRODUCER GAS

BY

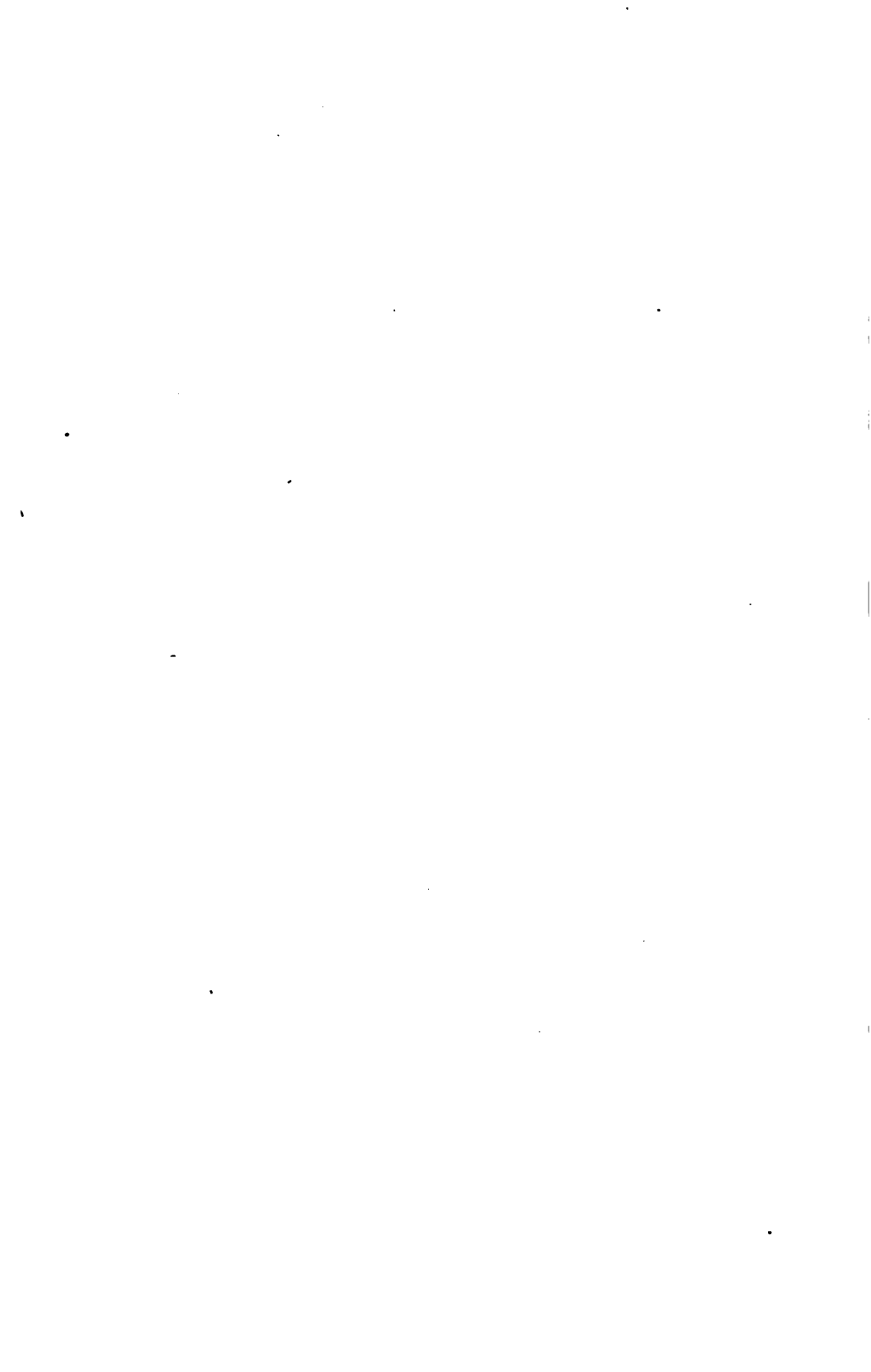
J. K. CLEMENT, L. H. ADAMS,
AND C. N. HASKINS



WASHINGTON

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ESSENTIAL FACTORS IN THE FORMATION OF PRODUCER GAS.

THE RATE OF FORMATION OF CARBON MONOXIDE AT HIGH TEMPERATURES.

By J. K. CLEMENT, L. H. ADAMS, and C. N. HASKINS.

EXPERIMENTAL INVESTIGATIONS.

By J. K. CLEMENT and L. H. ADAMS.

INTRODUCTION.

SCOPE AND PURPOSE OF INQUIRY.

In the course of its investigations of the fuel resources in the United States and of the methods by which these resources could be utilized with greatest efficiency, the United States Geological Survey tested a great variety of coals and lignites as gas-producer fuel. Early in these tests it was found that many factors controlling the formation of gas in the generating chamber of the producer and, consequently, having a direct bearing on the management of producer-gas plants, were not as well understood as they should be. The Geological Survey therefore took up a detailed investigation of the chemical and physical processes that take place in the producer, keeping in view not only the possibility of increasing the efficiency of the producer as a source of energy, and the ensuing benefits to the public of cheaper power and greater utilization of low-grade fuels, but also the application of the results to the problems of boiler-furnace operations.

The Bureau of Mines, to which the testing and analyzing of fuels as carried on by the United States Geological Survey has been transferred, is continuing producer-gas investigations at the testing station at Pittsburg, Pa. Results of the gas-producer tests^a made at the coal-testing plant erected at St. Louis, Mo., and of a study of some of the problems^b that came up in the tests have been published

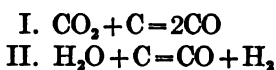
^a U. S. Geol. Survey Bull. Nos. 261, 290, 332, and Prof. Paper No. 48.

^b U. S. Geol. Survey Bull. No. 333.

by the Geological Survey. Results of the tests made at Norfolk, Va., and Pittsburg, Pa., and of further studies of particular problems, will be published by the Bureau of Mines.

In the gas producer solid fuel is transformed into more readily combustible gaseous fuel. The transformation is relatively slow and consists of several processes:

1. The distillation of the volatile hydrocarbons from the freshly fired fuel at relatively low temperatures.
2. The combustion of the fuel by its combining with the oxygen of the air forming carbon dioxide (CO_2).
3. The formation of carbon monoxide (CO) and hydrogen (H_2), the essential constituents of producer gas, in accordance with the equations:



The reactions represented by these two equations are most important in the process of producer-gas generation, and it is with the first reaction that this chapter has to do.

In experiments made by one of the writers^a at the fuel-testing plant of the United States Geological Survey at Norfolk, Va., it was found that the temperature in the fuel bed of the gas producer varied greatly in different parts of the bed. In order to ascertain the conditions of temperature most favorable to the efficient operation of the producer, it became necessary to determine the temperature required for the formation of carbon monoxide and hydrogen in accordance with the above reactions.

Another reason for investigating the conditions for the reduction of carbon dioxide by carbon was that a small quantity of carbon monoxide is invariably contained in the flue gases of boiler furnaces and it was hoped that a means might be suggested of preventing its formation and the resulting loss in furnace efficiency.

BOUDOUARD'S INVESTIGATIONS.

An elaborate series of determinations of the amount of CO formed at different temperatures was made by O. Boudouard.^b The observations were made at temperatures of 650° , 800° , and 925° C. In his experiments at 650° and 800° C. glass tubes containing charcoal, coke, retort carbon, or lampblack were filled with CO_2 , heated to the desired temperature, and then sealed. The tubes were maintained at constant temperature until equilibrium was reached;

^a Clement, J. K., and Grine, H. A., U. S. Geol. Survey Bull. No. 393, 1909, pp. 15-27.

^b Boudouard, O., Comptes rendus de l'Académie des Sciences, vol. 128, 1899, pp. 824, 1524; vol. 130, 1900, p. 132; vol. 131, 1900, p. 1204. Bulletin Soc. Chim., Paris, vol. 21, 1900; vol. 25, 1901; Ann. de Chimie et de physique, ser. 7, vol. 24, p. 354, 1901. See also Haber, Fritz, Thermodynamics of technical gas reactions, 1906, p. 811.

that is, until further heating at the same temperature produced no increase in the percentage of CO present. At 650° C. the heating was continued for 12 hours before equilibrium was attained. At 800° C. equilibrium was reached in 1 hour in the tubes containing charcoal and in 2½ hours in those containing lampblack. With coke and retort carbon the process was not complete at the end of 9 hours. In the experiment at 925° C. the carbon was heated in a porcelain tube, through which was passed a stream of CO₂. The average time of contact between CO₂ and carbon, calculated from the data given in Boudouard's account of his experiments, was approximately 30 seconds.

A summary of Boudouard's results is given in the following table:

Results of experiments by Boudouard on the formation of CO.

Temperature. (° C.).	CO ₂ (per cent.).	CO (per cent.).
650	61	39
800	7	93
925	4	96

These values have been made the basis of computations by many writers on the chemistry of combustion and the water-gas reaction, and have been especially prominent in treatises on the gas producer.

In the first references^a to Boudouard's work which came to the attention of the writers, no reference was made to the remarkably low reaction velocity of the formation of CO from CO₂ and carbon, and of the great length of time required to obtain the percentages of CO which are given in the above table. In at least one case the values given in the table were offered as representing the quality of the gas that should be obtained in the gas producer at the temperatures stated.^b The writers were, therefore, at first led to regard Boudouard's figures as defining the relative proportions of CO₂ and CO which should be formed in a gas producer at the various temperatures of the fuel bed, but such a view is quite erroneous, as the following experiments show:

EXPERIMENTS BY THE AUTHORS.

OBJECT OF EXPERIMENTS.

The experiments which form the theme of this chapter had originally as their object a critical repetition of Boudouard's experiments, and a continuation of them at higher temperatures. Preliminary

^a Haber, Fritz, *Thermodynamik Technischer Gas Reaktionen*. Munich and Berlin, 1905, p. 293. Fuchs, Paul, *Generator-, Kraftgas-, und Dampfkessel-Betrieb in Bezug auf Wärme-erzeugung und Wärme-verwendung*, 2nd edition, 1905.

^b Fuchs, loc. cit.

experiments, made by C. S. Hudson, demonstrated that the amount of CO formed at a given temperature depends largely upon the time of contact, or, in other words, on the rate of flow of the gas through the fuel bed. Apparently Boudouard's results represented the limiting values, which could be obtained only with a very low gas velocity. In order to ascertain the conditions for the formation of CO in producer furnaces, it was, therefore, necessary to determine the rate of formation of CO from CO_2 and carbon at various temperatures; that is, to determine the amount of CO formed with different rates of flow of the gas through the fuel bed.

APPARATUS.

Reaction tube and electric furnace.—The arrangement of the apparatus is shown in Plate I, and details of construction are given in figure 1 and figure 2. A porcelain tube of 1.5 cm. inside

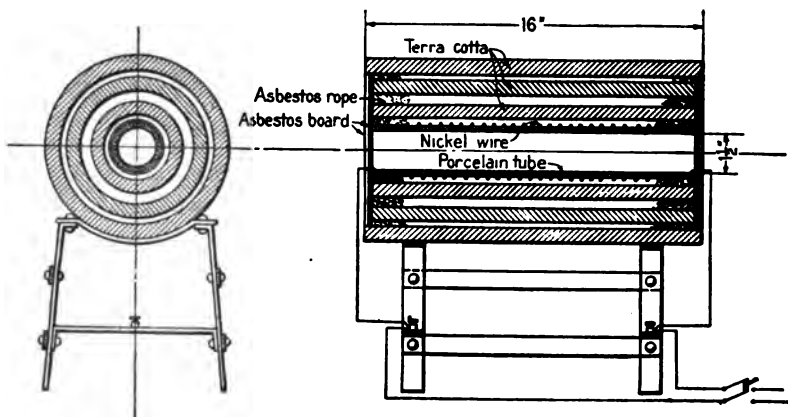


FIGURE 1.—Longitudinal and cross sections of electric furnace.

diameter, 60 cm. long, and glazed on the outside, was filled with charcoal, coal, or coke, and heated in an electric furnace (Plate I), which was designed especially for this investigation, and is shown in detail in figure 2. This furnace has been operated continuously

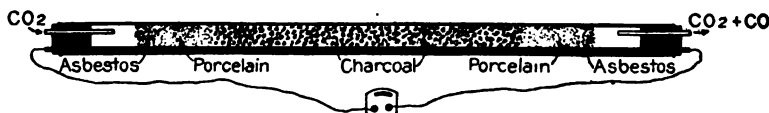
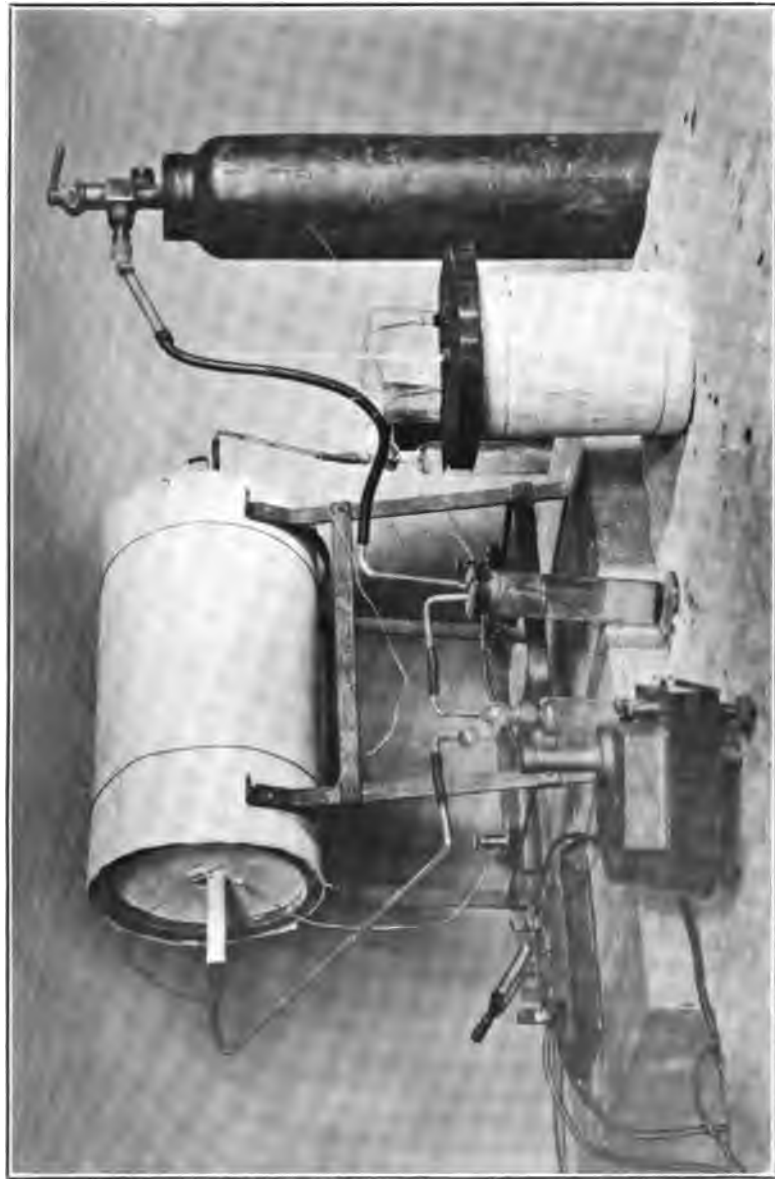


FIGURE 2.—Longitudinal section of reaction tube.

for a period of 6 months at temperatures of from 800° to $1,200^{\circ}$ C. and of even $1,300^{\circ}$ C. The temperature inside the porcelain tube can be maintained at any value up to $1,300^{\circ}$ C. with no fluctuations greater than 1° or 2° .

The heating coil is of No. 13 nickel wire, wound with eight turns per inch on an electrical porcelain insulating tube of 38 mm. inside



ELECTRIC FURNACE AND ACCESSORIES.

diameter and 35 cm. long. At either end of the coil the number of turns was increased slightly to compensate for the cooling effect at the end of the furnace. As a protection against corrosion the coil was painted with a thin layer of magnesite cement, a material capable of withstanding very high temperatures. The heating tube was then mounted inside of and concentric with three terracotta pipes, the spaces between the pipes having been filled with light calcined magnesia.

The cost of the materials used in the furnace and the amount of labor required were very small. The energy required to maintain a temperature of $1,000^{\circ}\text{C.}$ was 600 watts.

Thermocouples.—The temperature inside the porcelain tube was measured by means of a platinum, platinum-rhodium thermocouple (see fig. 2), and a Siemens & Halske millivoltmeter. As the thermoelectric height of the couple fell slightly with use at high temperatures, probably because of the reducing action of CO on the insulating tubes and the consequent contamination of the couple, it was found necessary to calibrate the couple from time to time. This was done by determining the melting points of zinc, silver, and copper. The error of individual temperature observations does not exceed 5° below $1,100^{\circ}\text{C.}$ nor 10° to 15° between $1,100^{\circ}$ and $1,300^{\circ}\text{C.}$

METHODS.

The carbon with which the porcelain tube was partly filled was crushed to pieces of a uniform size of about 5 mm. Only the middle part of the tube (see fig. 2) contained carbon, the remainder of the space being occupied by pieces of broken porcelain, which at one end served to heat the gas entering the tube, and at the other to increase the velocity of the gas through the region of falling temperature by reducing the size of the passageway. Through the porcelain tube was passed a stream of CO_2 . In the earlier experiments CO_2 was prepared from marble and hydrochloric acid; later it was taken from a tank of liquid CO_2 .

The velocity of the gas over the carbon was determined by the dimensions of the tube, the weight and density of the carbon, the temperature, and the volume of gas passed through the tube per minute.

The increase in the volume of the gas, due to the formation of two CO molecules in place of every molecule of CO_2 , which disappeared in the passage of the gas through the reaction tube, made it difficult to determine accurately the time of contact, and consequently the velocity of the gas.

The values of t (the time of contact) given in the tables below are based on the volume of gas leaving the tube, and are therefore somewhat too low. Since most of the expansion takes place within a

short distance from the entrance to the tube, the error introduced is probably not appreciable.

The analyses were made by the Hempel method, both CO_2 and CO being absorbed. The amount of gas remaining in the burette after the absorption in cuprous chloride was seldom greater than 2 per cent.

EXPERIMENTS WITH CHARCOAL.

With the apparatus described experiments were conducted at temperatures ranging from 700° to $1,300^\circ \text{C}$. The results of the observations with charcoal are contained in Table 1. The first and second columns of this table give the time of contact, t , and the reciprocal of the time of contact, $\frac{1}{t}$, which is proportional to the velocity of the gas through the fuel bed; that is,

$$\frac{1}{t} = \frac{\text{velocity of gas}}{\text{length of charcoal column}}$$

The third and fourth columns, respectively, give the quantity of CO , as observed and as calculated, in terms of percentage volume divided by 100. The values given for k_1 and k_2 are the respective velocity coefficients, at the particular temperature given, of the opposed reactions represented by the equations $\text{CO}_2 + \text{C} = 2 \text{CO}$ and $2 \text{CO} = \text{C} + \text{CO}_2$. For the method of calculation employed, the reader is referred to "Discussion of physical-chemical principles" (pp. 19–38).

A comparison of the results in Table 1, which are illustrated graphically in figures 3 and 4, shows, first, that with increasing temperature there is a rapid increase in the percentage of CO obtained with any given rate of flow of gas; second, that with increasing gas velocity the percentage of CO falls off very rapidly at low temperatures and very slowly at high temperatures. These variations are illustrated by curves in figure 3, in which the percentage of CO is plotted as a function of $\frac{1}{t}$.

TABLE 1.—Rate of formation of CO from CO_2 and charcoal.

AT A TEMPERATURE OF 800°C .

[$k_1 = 0.01968$. $k_2 = 3.031$.]

Time of contact, in seconds, t .	$\frac{1}{t}$	Per cent CO 100 observed.	Per cent CO 100 calculated.
∞	0	0.535
188.6	0.0053	0.503	0.534
115.9	0.0086	0.504	0.527
57.18	0.0175	0.518	0.508
45.70	0.0219	0.522	0.498
24.20	0.0413	0.375	0.545
15.50	0.0645	0.235	0.282
12.32	0.0810	0.245	0.209
2.686	0.354	0.063	0.051
1.550	0.645	0.039	0.030

TABLE 1.—Rate of formation of CO from CO₂ and charcoal—Continued.

AT A TEMPERATURE OF 850° C.

[$k_1=0.07174$. $k_2=3.238$.]

Time of contact, in seconds, t .	$\frac{1}{t}$	Per cent CO 100 observed.	Per cent CO 100 calculated.
∞	0	0.742
123.0	0.0082	0.743	0.742
54.1	0.0185	0.702	0.741
24.43	0.0410	0.572	0.694
13.23	0.0756	0.526	0.564
9.268	0.1070	0.287	0.463
4.630	0.216	0.287	0.281
3.686	0.271	0.224	0.231
3.254	0.307	0.225	0.207

AT A TEMPERATURE OF 900° C.

[$k_1=0.1540$. $k_2=2.599$.]

∞	0	0.873
64.29	0.0156	0.873	0.873
44.18	0.0226	0.867	0.872
10.008	0.0999	0.708	0.739
4.257	0.234	0.493	0.472
2.840	0.352	0.311	0.351
2.172	0.461	0.344	0.284

AT A TEMPERATURE OF 925° C.

[$k_1=0.2175$. $k_2=2.298$.]

∞	0	0.914
118.8	0.0084	0.947	0.914
81.2	0.0123	0.933	0.914
12.37	0.0807	0.848	0.875
5.80	0.1725	0.718	0.697
4.277	0.234	0.642	0.595
2.272	0.440	0.375	0.387

AT A TEMPERATURE OF 1,000° C.

[$k_1=0.6404$. $k_2=4.708$.]

∞	0	0.942
70.0	0.0143	0.949	0.942
18.60	0.0538	0.943	0.941
8.245	0.1195	0.908	0.938
3.675	0.272	0.797	0.869
2.296	0.436	0.795	0.752

AT A TEMPERATURE OF 1,100° C.

[$k_1=1.495$. $k_2=5.275$.]

∞	0	0.972
36.48	0.0274	0.967	0.972
10.43	0.0958	0.963	0.972
4.968	0.2010	0.961	0.971
3.640	0.2745	0.973	0.968
1.921	0.521	0.946	0.955

Since $\frac{1}{t} = \frac{v}{l}$, when l , the length of the charcoal column, is equal to 1, that is, equal to the unit of length, then the numbers along the $\frac{1}{t}$ -axis give the velocity of gas in terms of the same unit of length and

seconds. For example, the length of the charcoal column in the experiments here recorded was approximately 25 cm., or 10 inches. The velocity corresponding to the point $\frac{1}{t} = 1$, at the extreme right of figure 3, is, therefore, 10 inches per second.

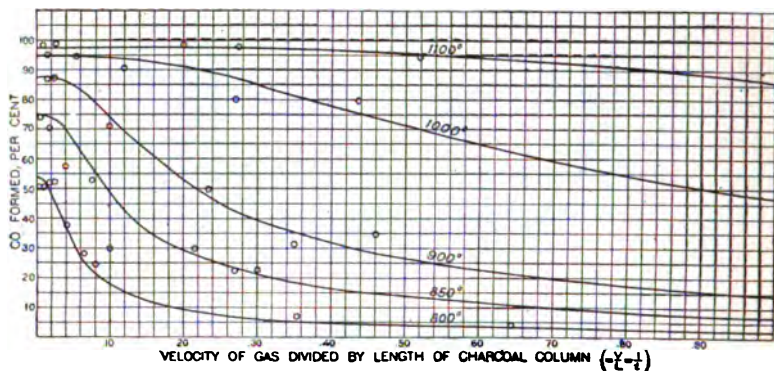


FIGURE 3.—Variation of percentage of CO formed from CO_2 and charcoal with change of gas velocity.

The curves through the points in the figure have not been arbitrarily drawn, but have been plotted from a mathematical equation, which expresses the percentage of CO as a function of $\frac{1}{t}$. (See "Discussion of physical-chemical principles.") The general shape of all the

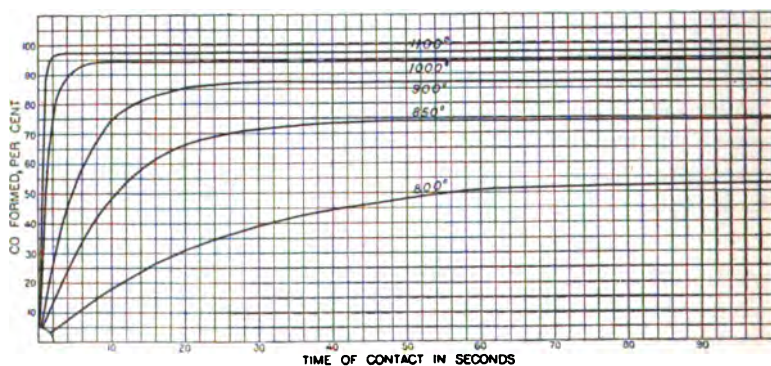


FIGURE 4.—Variation of percentage of CO formed from CO_2 and charcoal with change of time of contact.

curves in figure 3 is the same. The percentage of CO is greatest at zero velocity ($\frac{1}{t} = 0, t = \infty$). With increasing values of $\frac{1}{t}$, each curve falls away at first slowly, then more rapidly, passing a point of inflection, and, finally, becoming nearly horizontal. The intersections of the curves with the CO axis give the percentages of CO corresponding to the condition of equilibrium.

That a considerable amount of time is required to reach equilibrium in the reaction under consideration is further illustrated in figure 4, in which the percentage of CO is platted as a function of t , the time of contact. At 800°C ., for example, the percentage of CO reaches a practically constant value at the end of 50 seconds; at $1,000^{\circ}\text{C}$. in 6 seconds.

In figure 3 (p. 12) the curve for 800°C . falls off very rapidly with increasing rate of flow of gas. At this temperature the gas velocity must be exceedingly low to obtain the equilibrium percentage of CO. At temperatures below 800°C . equilibrium could not be reached with the lowest gas velocities which could be maintained. A great number of experiments were made at 700°C ., but the results were too in-

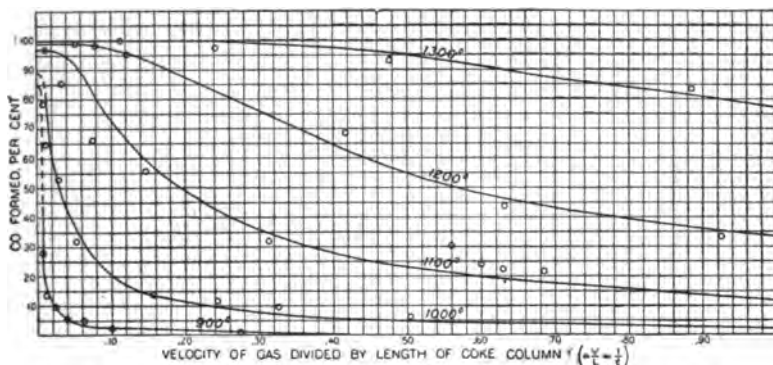


FIGURE 5.—Variation of percentage of CO formed from CO_2 and coke with change of gas velocity.

consistent to admit of mathematical treatment. Some of the observations are given in the following table:

Observations on the formation of CO from CO_2 and charcoal at a temperature of 700°C .

Time of contact, in seconds, t .	$\frac{1}{t}$	Per cent CO 100
99.9	0.0100	0.077
96.9	0.0115	0.012
96.2	0.0116	0.155
23.4	0.0426	0.004
15.0	0.0668	0.014
7.11	0.1406	0.009
9.71	0.103	0.022
5.60	0.178	0.006
5.02	0.199	0.006
4.18	0.239	0.012

These results show that, except at exceedingly low velocities, the amount of CO formed was never greater than 1 or 2 per cent.

EXPERIMENTS WITH COKE AND WITH ANTHRACITE.

The experiments with coke and with anthracite were conducted in the same manner as with charcoal. Table 2 contains the results of the observations with coke, while in figure 5 the same results are shown graphically.

The curves for 900°, 1,000°, and 1,100° C. are considerably lower than those for charcoal at the same temperatures except for very low velocities.

TABLE 2.—Rate of formation of CO from CO₂ and coke.

AT A TEMPERATURE OF 900° C.

[$k_1=0.00231$. $k_2=0.03686$.]

Time of contact, in seconds, <i>t</i> .	$\frac{1}{t}$	Percent CO 100 observed.	Percent CO 100 calculated.
142.0	0.0070	0.276	0.278
80.20	0.0124	0.131	0.169
43.91	0.0228	0.094	0.096
24.82	0.0403	0.057	0.056
16.11	0.0620	0.049	0.037
9.575	0.1045	0.026	0.023
3.741	0.2671	0.008	0.009

AT A TEMPERATURE OF 1,000° C.

[$k_1=0.02323$. $k_2=0.3591$.]

123.2	0.0081	0.784	0.866
80.25	0.0125	0.644	0.795
33.25	0.0301	0.529	0.527
18.72	0.0535	0.320	0.350
6.37	0.1571	0.139	0.138
4.101	0.2439	0.115	0.091
3.072	0.3258	0.092	0.069
1.983	0.5045	0.063	0.045

AT A TEMPERATURE OF 1,100° C.

[$k_1=0.1335$. $k_2=0.5296$.]

90.00	0.0111	0.971	0.971
29.92	0.0334	0.854	0.955
13.20	0.0758	0.661	0.817
6.765	0.1476	0.556	0.592
3.198	0.3135	0.317	0.346
1.784	0.5606	0.304	0.211
1.660	0.6030	0.240	0.1942
1.590	0.6299	0.221	0.1190
1.462	0.6840	0.214	0.177
0.962	1.0399	0.133	0.121

AT A TEMPERATURE OF 1,200° C.

[$k_1=0.4095$. $k_2=0.6718$.]

18.92	0.0528	0.989	0.987
12.70	0.0788	0.978	0.983
8.250	0.1213	0.953	0.956
2.402	0.4160	0.685	0.624
1.582	0.6320	0.439	0.460
1.080	0.9260	0.335	0.357

AT A TEMPERATURE OF 1,300° C.

[$k_1=1.483$. $k_2=0.7313$.]

8.860	0.1129	0.999	0.997
4.149	0.2415	0.979	0.997
2.100	0.4760	0.932	0.955
1.130	0.8850	0.834	0.816

TABLE 3.—Rate of formation of CO from CO₂ and anthracite.

AT A TEMPERATURE OF 1,100° C.

[$k_1=0.119$. $k_2=1.410$.]

Time of contact, in seconds, t .	$\frac{1}{t}$	Per cent CO $\frac{100}{\text{observed.}}$	Per cent CO $\frac{100}{\text{calculated.}}$
34.20	0.0293	0.8780	0.912
9.370	0.1069	0.6010	0.667
5.415	0.1848	0.4770	0.472
3.801	0.3026	0.3020	0.322
2.439	0.4101	0.2650	0.251

AT A TEMPERATURE OF 1,200° C.

[$k_1=0.2374$. $k_2=0.1767$.]

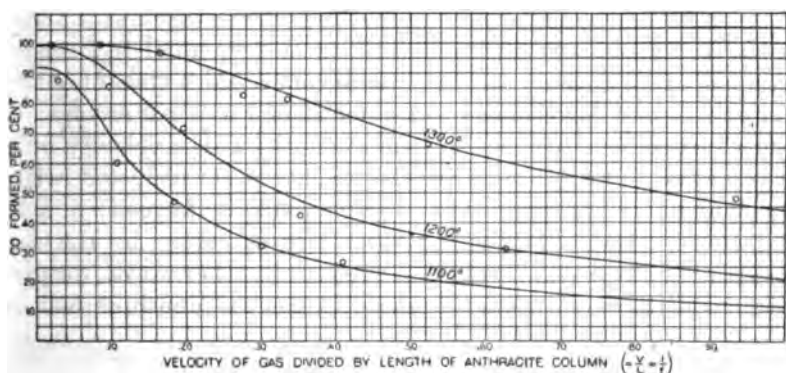
47.05	0.0212	0.997	0.993
10.39	0.0964	0.856	0.901
5.070	0.1971	0.715	0.688
2.845	0.3516	0.423	0.472
1.592	0.6270	0.310	0.309

AT A TEMPERATURE OF 1,300° C.

[$k_1=0.5791$. $k_2=0.2016$.]

12.40	0.0806	0.999	0.997
6.030	0.1659	0.965	0.968
3.600	0.2779	0.824	0.876
2.980	0.3358	0.809	0.822
1.908	0.5249	0.663	0.668
1.070	0.9350	0.503	0.462

The results of observations with anthracite are given in Table 3 and are illustrated graphically in figure 6. Here the curves fall off even more rapidly than those for coke in figure 5.

FIGURE 6.—Variation of percentage of CO formed from CO₂ and anthracite with change of gas velocity.

With very low velocities—that is, when the time of contact is sufficient for the reaction to reach equilibrium—the percentage of CO formed is practically the same with each of the three forms of carbon. The effect of the differences in the reaction velocities becomes more appreciable as the rate of flow of gas increases.

APPLICATION OF RESULTS OF EXPERIMENTS TO GAS-PRODUCER AND BOILER-FURNACE PROCESSES.

As stated in the introduction, the experiments here described were undertaken primarily to determine the temperature necessary for the formation of a gas containing a high percentage of CO in the fuel bed of the gas producer and to ascertain the conditions which govern the formation of CO in boiler furnaces.

FACTORS GOVERNING THE FORMATION OF CARBON MONOXIDE.

The results which are here presented indicate that the amount of CO formed in the gas producer depends on three factors—(1) the temperature, (2) the depth of the hot portion of the bed, and (3) the rate of flow of gas through the bed. To state it more concisely, the percentage of CO formed depends on the temperature and the time of contact of gas and carbon—that is, the average time required for a molecule of gas to pass through the fuel bed. The variation of the percentage of CO, with the rate of flow of gas, is illustrated in figures 3, 5, and 6. The curves for coke (fig. 5) may be taken as representing the conditions in the fuel bed of the producer. At 1,300° C., for example, with zero velocity (time of contact = ∞), practically all the CO₂ will be converted to CO; when $\frac{1}{t} = 0.5$ (time of contact, $t = 2$ seconds), 90 per cent CO is obtained; and when $t = 1$, only 80 per cent CO is formed. Since

$$\frac{1}{t} = \frac{\text{velocity of gas}}{\text{depth of fuel bed}} = \frac{v}{l}$$

in a fuel bed 1 foot in depth a time of contact, t , equal to 2 seconds corresponds to a velocity 0.5 foot per second, and a time of contact equal to 1 second corresponds to a velocity of 1 foot per second. At 1,300° C., then, in a fuel bed 1 foot in depth, with a velocity of 0.5 foot per second, 90 per cent of CO would be formed, and with a velocity of 1 foot per second, 80 per cent would be formed; in a fuel bed 2 feet in depth the gas velocities corresponding to the same percentage of CO would be twice as great. In other words, for given conditions of temperature and quality of gas the depth of bed and velocity of gas must vary proportionally and their ratios $\frac{v}{l}$ must remain constant. A fuel bed 1 foot in depth and a gas velocity of 1 foot per second should yield the same percentage of CO as a bed 2 feet in depth with a gas velocity of 2 feet per second.

It is impossible to determine accurately the velocity of the gas through the producer fuel bed, because of the difficulty of estimat-

ing the magnitude of the passages through the bed.^a The velocity is probably between 0.5 and 5 feet per second. The right half of the curves in figure 5 lies within these limits, and therefore corresponds, approximately, to the conditions of producer operation.

Figure 7 shows graphically the variation in the amount of CO formed with rise of temperature at various values of $\frac{1}{t}$. The ordinate is the percentage of CO in gas initially containing 21 per cent CO₂ (air in which the oxygen has been converted quantitatively to CO₂). The abscissa is temperature in degrees centigrade. The upper curve for which $\frac{1}{t}=0$ represents the maximum amount of CO which could be produced from air. The intersection of the curve for any velocity with a given horizontal line—for example, the line for CO=30 per cent—gives the temperature required to form that

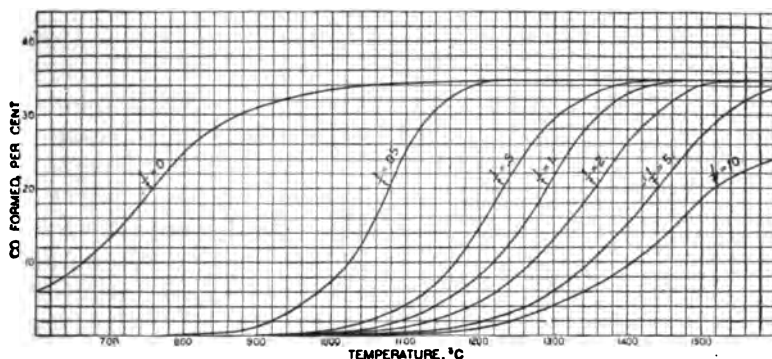


FIGURE 7.—Variation of percentage of CO formed with rise of temperature, using air and coke.

amount of CO with the particular velocity. Thus, to obtain 30 per cent CO with a velocity of 1 foot per second (depth of bed=1 foot) will require a temperature of 1,360° C., and with a velocity of 2 feet per second, 1,435° C. The curves of figures 5 and 7 indicate that the temperature of the producer bed should not be less than 1,300° C.

DISADVANTAGE OF AN EXTREMELY HOT FUEL BED.

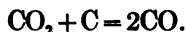
These investigations demonstrate that a very high temperature is necessary for the production of CO from CO₂ and carbon. There are considerations, however, which argue against operating the fuel bed of the gas producer at extremely high temperatures—above 1,300° C.

^a When any given number of pounds of air per second is passed through a fuel bed of given dimensions, the velocity through the bed will increase as the percentage of voids is decreased; e. g., the velocity will be much higher with slack coal than with uniformly sized nut coal. Further, the percentage of voids will be influenced by the amount of coking and clinkering.

A very hot fuel bed means that the gases will leave the producer at a high temperature, and thus lower the efficiency of the producer. The gain in capacity would, therefore, be accompanied by a loss in efficiency, unless the heat of the gases could be used for generating steam or preheating the air blast. A high temperature also favors clinkering. In the application of the results of these experiments to commercial producers and furnaces it will be necessary, of course, to consider the other questions which are involved.

CARBON MONOXIDE IN FLUE GAS FROM BOILER FURNACES.

Various explanations have been suggested to account for the presence of small amounts of CO in the flue gases of boiler furnaces. Perhaps the one most generally accepted by engineers is that the oxygen of the air first unites with carbon to form CO_2 , and that as this gas passes up through the hot fuel bed it combines with carbon in accordance with the equation:



Assuming this to be the correct explanation, then the question to be solved is what conditions are favorable to this reaction and what conditions will tend to retard it. In the preceding paragraphs it has been shown that the higher the velocity of the gas and the thinner the fuel bed the less will be the percentage of CO formed. A heavy fuel bed in the boiler furnace would therefore favor the formation of CO. Also, the greater the supply of air to a given depth of bed the less should be the percentage of CO formed. These results show that with a hot fuel bed the formation of a small amount of CO is inevitable. In order that this CO may be burned to CO_2 , it is necessary that in some way sufficient air be added to the hot gases as they leave the top of the fuel bed.

DISCUSSION OF PHYSICAL-CHEMICAL PRINCIPLES.

By J. K. CLEMENT, L. H. ADAMS, and C. N. HASKINS.

APPLICATION OF THE LAWS OF CHEMICAL EQUILIBRIUM AND REACTION KINETICS.

The velocity of a reaction in a homogeneous system is at any moment defined by the velocity coefficients and the concentrations of the various substances taking part in the reaction.

In heterogeneous systems, however, the reaction velocity depends usually on the rate of diffusion of the reacting substances, and on the area of the boundary surface between the two phases.^a

Nernst's^b theory, that in the boundary layer between a solid and a liquid phase equilibrium is established almost instantaneously and that the reaction velocity depends solely on the rate of diffusion, has been confirmed by Brunner^c in several cases. But, as Haber^d points out, the reactions which Brunner studied "involve merely the addition of a charge to substances going over into the ionic condition, and the loss of a charge by ions leaving the ionic condition."

Such changes universally take place instantaneously. Nernst's theory of the velocities of heterogeneous reactions applies when the chemical-reaction velocities are great compared with the rate of diffusion, but not necessarily when the reverse relation is met with.

Since the rate of diffusion of gases as compared to liquids is enormous, it seems quite possible that in reactions between solids and gases the rate of reaction may depend on the velocity of the chemical combination, as well as on the rate of diffusion. It is easily conceivable that the rate of diffusion may become so great compared with the reaction velocity that the latter becomes the determining factor. In this event the laws of reaction velocity which have been derived for homogeneous systems may be applied to heterogeneous reactions.

In the absence of any knowledge regarding the relative magnitudes of chemical-reaction velocity and rate of diffusion in any given

^a Boguski, *Ber. deutsch. chem. Ges.*, vol. 9 (1876), p. 1646. Noyes and Whitney, *Zeitschr. physik. Chem.*, vol. 23 (1897), p. 689.

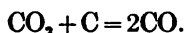
^b Nernst, Walther, *Theoretische Chem.*, 5th ed. (Stuttgart, 1907), pp. 578, 579; and *Zeitschr. f. physik. Chem.*, vol. 47 (1904), p. 52.

^c Brunner Karl, *Zeitschr. physik. Chem.* vol. 47 (1904), p. 56.

^d Haber, Fritz, *Thermodynamics of technical gas reactions*, London, 1906, pp. 253-254.

instance, the application of the laws of reaction velocity to a heterogeneous reaction can be justified only by the results obtained. If the velocity coefficients prove to be constant over a wide range of values of t (the time of contact), and if, further, the increase with temperature of the velocity coefficients is of the magnitude commonly observed in the case of homogeneous systems, then, in all probability, the determining factor in the reaction velocity is the velocity of chemical combination and not that of diffusion. The diffusion constants of gases vary according to the one and one-half to the second power of the absolute temperature.^a The latter value would correspond to an increase of 2.6 per cent for a rise of 10° at 500°C. , and of 1.6 per cent for the same rise of temperature at $1,000^\circ\text{C.}$ It will be shown later that the increase of coefficients of reaction velocity for the same rise of temperature should be about 20 per cent at 500°C. , and perhaps 10 per cent at $1,000^\circ\text{C.}$ ^b

The object of attempting to apply the laws of reaction kinetics to the results of the experiments described in "Experimental investigations" (pp. 7-15) was rather to obtain a means of interpolating and extrapolating the results than to furnish a confirmation of the laws themselves. The splendid agreement, however, between the observed and calculated values of the percentage of CO obtained with various gas velocities and the rapid increase of the rate of formation of CO with rise in temperature, argue in favor of the velocity of chemical combination as the determining factor in the speed of the reaction



THE EQUILIBRIUM CONSTANT.

It follows from the law of chemical mass action that in the heterogeneous system $\text{C} + \text{CO}_2 \rightleftharpoons 2\text{CO}$,^c for every temperature there is a certain constant relation between concentrations of CO and CO_2 in equilibrium with carbon.

In the system under consideration, thermodynamic reasoning leads to the conclusion that

$$\frac{[\text{CO}]^2}{[\text{CO}_2]} = \text{constant} = K,$$

in which [CO] and $[\text{CO}_2]$ denote the concentrations of CO and CO_2 , respectively, in gram molecules per liter.^d

^a See Meyer, O. E., *Kinetische Theorie der Gase*, 2d ed. (Breslau, 1899), sec. 101.

^b See Hoff, J. H. van't, *Chemical dynamics*, London, 1898, p. 228, and Haber, Fritz, *Thermodynamics of technical gas reactions*, p. 256.

^c The reverse arrows (\rightleftharpoons) used in place of the regular symbol ($=$) in the equation $\text{C} + \text{CO}_2 \rightleftharpoons 2\text{CO}$ denote that the reaction is reversible; that is, it may take place in the direction from left to right or from right to left.

^d A "gram molecule" of any substance is its molecular weight in grams; e. g., a gram molecule of H_2O is 18 grams, of CO, 28 grams, and of CO_2 , 44 grams.

Let $100x$ = percentage by volume of CO and $100(1-x)$ = percentage by volume of CO_2 ,

Then
$$[\text{CO}] = x \frac{P}{R\theta}$$

$$[\text{CO}_2] = (1-x) \frac{P}{R\theta}$$

and
$$\frac{[\text{CO}]^2}{[\text{CO}_2]} = \frac{x^2}{(1-x)} \frac{P}{R\theta} = K.$$

Here R = the gas constant = $\frac{22.4}{273} = 0.0821$ liter atmospheres,

P = the pressure in atmospheres,

and θ = the absolute temperature.

$[\text{CO}]$ = the number of gram molecules of CO per liter.

$[\text{CO}_2]$ = the number of gram molecules of CO_2 per liter.

For constant conditions of pressure and temperature,

$$\frac{P}{R\theta} \text{ is a constant and } \frac{x^2}{1-x} = \text{constant} = K \frac{R\theta}{P}$$

The significance of this equation is that if a quantity of CO_2 gas is maintained in contact with carbon at constant temperature, θ , and pressure, P , the gas and the carbon will react rapidly at first, and then more and more slowly until the amount of CO formed is exactly $100x$ per cent of the total gas present.

THE REACTION-VELOCITY EQUATION.

Dynamic considerations as well as thermodynamic reasoning lead to the equation

$$\frac{[\text{CO}]^2}{[\text{CO}_2]} = K.$$

In the former case K takes the value $\frac{k_1}{k_2}$, where the k 's are the velocity coefficients of the two opposed reactions. The equation representing the rate of formation of CO by the reaction $\text{CO}_2 + \text{C} \rightleftharpoons 2\text{CO}$ may be written as follows:

$$\frac{d[\text{CO}]}{dt} = k_1[\text{CO}_2] - k_2[\text{CO}]^2. \quad (1)$$

When equilibrium is established

$$k_1[\text{CO}_2] = k_2[\text{CO}]^2$$

and

$$\frac{[\text{CO}]^2}{[\text{CO}_2]} = \frac{k_1}{k_2} = K.$$

If $100a$ = percentage of CO_2 by volume at time $t = 0$,

$100x$ = percentage of CO by volume at time t ;

and when

$$t = 0, x = 0,$$

then when

$$t = 0, \quad [\text{CO}_2] = a \frac{P}{R\theta} = M.$$

Let n = total number of CO molecules at time t , and m = total number of CO_2 molecules at time t .

Then if the initial volume is unity, $m = M - \frac{n}{2}$, and the corresponding volume at constant pressure is $1 + \frac{n}{2} \frac{R\theta}{P}$.

Then,

$$[\text{CO}] = x \frac{P}{R\theta} = \frac{n}{1 + \frac{n}{2} \frac{R\theta}{P}}$$

$$n = \frac{2x}{2-x} \frac{P}{R\theta}$$

and

$$[\text{CO}_2] = \frac{m}{1 + \frac{n}{2} \frac{R\theta}{P}} = \frac{M - \frac{n}{2}}{1 + \frac{n}{2} \frac{R\theta}{P}} = \frac{a \frac{P}{R\theta} - \frac{n}{2}}{1 + \frac{n}{2} \frac{R\theta}{P}} = \frac{a \frac{P}{R\theta} - \frac{x}{2-x} \frac{P}{R\theta}}{1 + \frac{x}{2-x} \frac{P}{R\theta}} = \frac{a - \frac{x}{2-x}}{1 + \frac{x}{2-x}} \frac{P}{R\theta}.$$

That is,

$$[\text{CO}_2] = \left(a - \frac{a+1}{2} x \right) \frac{P}{R\theta}$$

Introducing the values in equation (1) we obtain

$$\frac{dx}{dt} = k_1 \left(a - \frac{a+1}{2} x \right) - k_2 \frac{P}{R\theta} x^2,$$

and if

$$k'_2 = k_2 \frac{P}{R\theta}$$

$$\frac{dx}{dt} = k_1 \left(a - \frac{a+1}{2} x \right) - k'_2 x^2 \quad (2)$$

For the case that the initial gas is pure CO_2 , $a = 1$, and

$$\frac{dx}{dt} = k_1(1-x) - k'_2 x^2.$$

The integration of the differential equation (2) offers no great difficulty. The determination of the constants k_1 and k'_2 , seems to have been made hitherto only by assuming the value of the ratio $\frac{k_1}{k'_2}$. This method is applicable when the equilibrium conditions are readily realized. As, however, this is not the case in the present reaction, it has been necessary to devise a method for the determination of k_1 and k_2 from any two or more pairs of simultaneous observation of x and t . It turns out that the method is applicable not only to the present reaction, but also to the general incomplete reaction of the second order.

The great difficulty of realizing, with certainty, the condition of the equilibrium in reactions like the one under consideration makes it highly desirable to obtain a general solution of the differential equation without introducing a particular numerical value of $\frac{k_1}{k'_2}$.

Prof. C. N. Haskins has obtained the solution of the differential equation (2) that is here presented.

1. Reduction and integration of the differential equation.

The differential equation is

$$\frac{dx}{dt} = k_1 \left(a - \frac{a+1}{2} x \right) - k'_2 x^2, \quad (2)$$

where $a = \frac{\text{per cent CO}_2}{100}$ at time $t=0$,

$x = \frac{\text{per cent CO}}{100}$ at time t ,

$t = \text{time in seconds}$,

and k_1 and k'_2 are the two constants of the reaction the values of which are sought. The initial condition is that

$$x=0, \text{ when } t=0.$$

To integrate, we introduce a new variable, z , and new constants, α and γ defined by the relations

$$\left. \begin{aligned} z &= \frac{x(a+1)}{4a-x(a+1)} & x &= \frac{4az}{(a+1)(z+1)} \\ \alpha &= \frac{1}{2} \sqrt{k_1^2(a+1)^2 + 16ak_1k'_2} & k_1 &= \frac{4\alpha\gamma}{a+1} \\ \gamma &= \sqrt{\frac{k_1(a+1)^2}{k_1(a+1)^2 + 16ak'_2}} & k'_2 &= \frac{\alpha(a+1)(1-\gamma^2)}{4a\gamma} \end{aligned} \right\} (3)$$

The differential equation becomes, under these substitutions

$$\frac{dz}{dt} = \frac{\alpha}{\gamma} (\gamma^2 - z^2) \quad (4)$$

with the initial conditions

$$z = 0, \text{ when } t = 0. \quad (5)$$

Integrating, we have

$$\ln^* \frac{\gamma + z}{\gamma - z} = 2\alpha t \quad (6)$$

and solving for z ,

$$z = \gamma \frac{e^{\alpha t} - e^{-\alpha t}}{e^{\alpha t} + e^{-\alpha t}} = \gamma \tanh \alpha t \quad (7)$$

whence, substituting in (3)

$$x = \frac{4a}{a+1} \frac{\gamma \tanh \alpha t}{1 + \gamma \tanh \alpha t} \quad (8)$$

2. *The equation for γ and the criterion for the existence of one and only one root.*

We have (equation 8) an expression by means of which the percentage of CO at any time, t , may be computed if the constants γ and α are known. We now wish to determine γ and α from two pairs of observed corresponding values of t and x . Let these two pairs be t_1, x_1 and t_2, x_2 , and let $t_2 > t_1$. Then since a is known, we may compute

$$z_1 = \frac{x_1(a+1)}{4a - x_1(a+1)}, \quad z_2 = \frac{x_2(a+1)}{4a - x_2(a+1)}$$

and have

$$\ln \frac{\gamma + z_1}{\gamma - z_1} = 2 \alpha t_1, \quad \ln \frac{\gamma + z_2}{\gamma - z_2} = 2 \alpha t_2$$

from which γ and α are to be determined. Eliminating, α we readily obtain

$$\ln \frac{\gamma + z_2}{\gamma - z_2} = \frac{t_2}{t_1} \ln \frac{\gamma + z_1}{\gamma - z_1} \quad (9)$$

The determination of γ and α depends therefore on the solution of this (transcendental) equation in γ . Consideration of the function

$$U(\gamma) \equiv t_1 \ln \frac{\gamma + z_2}{\gamma - z_2} - t_2 \ln \frac{\gamma + z_1}{\gamma - z_1}$$

and of its derivative

$$U'(\gamma) \equiv -2 \left(\frac{t_1 z_2}{\gamma^2 - z_2^2} - \frac{t_2 z_1}{\gamma^2 - z_1^2} \right)$$

* The symbols $\ln x$, $\log x$, will be used to denote the natural and the common logarithm of x , respectively.

shows that the equation

$$\ln \frac{\gamma+z_2}{\gamma-z_2} = \frac{t_2}{t_1} \ln \frac{\gamma+z_1}{\gamma-z_1} \quad (9)$$

has a root $\gamma > z_2$ when and only when

$$t_2 z_1 - t_1 z_2 > 0, \text{ that is, } \frac{z_2}{z_1} < \frac{t_2}{t_1} \quad (10)$$

If a root exists there is but one, and it satisfies the inequalities

$$z_2 < \gamma < \sqrt{z_1 z_2 \frac{(t_2 z_2 - t_1 z_1)}{t_2 z_1 - t_1 z_2}}.$$

The inequality (10) furnishes a negative criterion for the applicability of the differential equation (2) to a reaction under investigation. For if the reaction is governed by that equation, and if the observations are made with sufficient accuracy, there must exist a γ satisfying equation (9) and hence the inequality (10) must be satisfied. If then this inequality is not satisfied, and hence no such γ can be found, the assumptions involved in (2) must be invalid, or there must be error in the observations. On the other hand, if (10) is satisfied we can only conclude that (2) *may* be applicable, and we proceed to determine whether it is so by computing γ and α and comparing the values of x computed by means of (8) with the observed values of x .

3. Solution of the equation

$$\ln \frac{\gamma+z_2}{\gamma-z_2} = \frac{t_2}{t_1} \ln \frac{\gamma+z_1}{\gamma-z_1}.$$

If the selected pairs $t_1 x_1$, $t_2 x_2$ of observed values satisfy the criterion

$$\frac{z_2}{z_1} < \frac{t_2}{t_1} \quad (10)$$

we may compute γ in the equation as follows: Passing, for convenience of computation, from natural to common logarithms we have

$$\log \frac{\gamma+z_2}{\gamma-z_2} = \frac{t_2}{t_1} \log \frac{\gamma+z_1}{\gamma-z_1}.$$

Assume now a value for γ , say $\gamma = \gamma_1$, where

$$z_2 < \gamma < \sqrt{z_1 z_2 \frac{(t_2 z_2 - t_1 z_1)}{t_2 z_1 - t_1 z_2}}.$$

Then we may compute a quantity $\log N_1$ by the equation

$$\log N_1 = \frac{t_2}{t_1} \log \frac{\gamma+z_1}{\gamma-z_1}.$$

Determine now a new value of γ , $\gamma = \gamma_2$ by the relation

$$\log \frac{\gamma_2 + z_2}{\gamma_2 - z_2} = \log N_1$$

that is,

$$\frac{\gamma_2 + z_2}{\gamma_2 - z_2} = N_1$$

or ^a

$$\gamma_2 = \frac{N_1 + 1}{N_1 - 1} z_2.$$

Proceed now to determine a new approximation γ_3 from γ_2 in the same way that γ_2 was determined from γ_1 and continue the process until its repetition produces no change, or a change which is negligible compared with the experimental errors. It will be found that in general the process converges somewhat rapidly, and only a few repetitions are necessary.

Suppose, then, that γ has been found by this process. Then α is computed by either of the relations

$$2\alpha = \frac{1}{t_1} \ln \frac{\gamma + z_1}{\gamma - z_1}, \quad 2\alpha = \frac{1}{t_2} \ln \frac{\gamma + z_2}{\gamma - z_2}$$

or, what amounts to the same thing, if N is the last of the numbers N_1, N_2, \dots used in computing γ

$$2\alpha = \frac{1}{t_2} \ln \frac{\gamma + z_2}{\gamma - z_2} = \frac{1}{t_2} \ln N = \frac{1}{t_2} \log N \ln 10$$

$$2\alpha = \frac{2.3026}{t_2} \log N.$$

4. Computation of reaction constants k_1 , k'_2 , and verification of reaction equation.

When the constants α , γ have been computed we can find the original constants k_1 , k'_2 by the relations

$$k_1 = \frac{4\alpha\gamma}{a+1}$$

$$k'_2 = \frac{\alpha(a+1)(1-\gamma^2)}{4a\gamma} \quad (3)$$

To determine the applicability of the reaction equation (2) to the case in hand we have now only to introduce the values just found into the equation

$$z = \gamma \tanh \alpha t \quad (7)$$

or

$$x = \frac{4a}{a+1} \frac{\gamma \tanh \alpha t}{1 + \gamma \tanh \alpha t} \quad (8)$$

^aThis computation may be abridged by the use of Gaussian logarithms.

and compare the values of z or x given with those found by observation. For this purpose equation (7) is the simpler, especially when the z 's corresponding to observed values of x have been already computed.

5. *Correction of constants α and γ by the method of least squares.*

The constants α , γ obtained above are determined by two pairs of observations only. It is, of course, desirable that all the observations be used in fixing their values, as on account of experimental errors the values obtained from different pairs of observations will in general not be identical. We proceed, therefore, to correct the constants by the method of least squares.

Let $t_1, x_1; t_2, x_2; \dots t_n, x_n$, be the n observed pairs of values of t and x . The problem is, then, to determine α and γ in such a way that if

$$\ln \frac{\gamma + z_i}{\gamma - z_i} - 2\alpha t_i \equiv \delta_i$$

then

$$n\delta^2 \equiv \sum_1^n \delta_i^2$$

shall be a minimum.

In order that α and γ shall make $n\delta^2$ a minimum they must satisfy the so-called normal equations.

$$\frac{n}{2} \frac{d\delta^2}{d\alpha} \equiv \sum_1^n \delta_i \frac{d\delta_i}{d\alpha} \equiv \sum_1^n 2t_i \left\{ 2\alpha t_i - \ln \frac{\gamma + z_i}{\gamma - z_i} \right\} = 0,$$

$$\frac{n}{2} \frac{d\delta^2}{d\gamma} \equiv \sum_1^n \delta_i \frac{d\delta_i}{d\gamma} \equiv \sum_1^n t_i \left\{ \frac{1}{\gamma + z_i} - \frac{1}{\gamma - z_i} \right\} \left\{ \ln \frac{\gamma + z_i}{\gamma - z_i} - 2\alpha t_i \right\} = 0$$

These equations may be written

$$2\alpha \sum_1^n t_i^2 + \sum_1^n t_i \ln \frac{\gamma - z_i}{\gamma + z_i} = 0.$$

$$2\alpha \sum_1^n t_i \frac{t_i z_i}{\gamma^2 - z_i^2} + \sum_1^n t_i \frac{z_i}{\gamma^2 - z_i^2} \ln \frac{\gamma - z_i}{\gamma + z_i} = 0$$

As their exact solution in their present form is impracticable on account of their complexity, we replace them in the usual way by a system of approximately equivalent linear equations, by making use of the fact that the quantities u , v by which α and γ differ from α_0 and γ_0 respectively are small compared with α_0 , γ_0 .

Substituting

$$\alpha = \alpha_0 + u$$

$$\gamma = \gamma_0 + v$$

we have

$$\begin{aligned} 2u \sum_1^n t_i^2 + \sum_1^n t_i \ln \left(\frac{1 + \frac{v}{\gamma_0 - z_i}}{1 + \frac{v}{\gamma_0 + z_i}} \right) &= \sum_1^n t_i \left\{ \ln \frac{\gamma_0 + z_i}{\gamma_0 - z_i} - 2\alpha_0 t_i \right\} \\ 2u \sum_1^n t_i \frac{t_i z_i}{(\gamma_0^2 - z_i^2) \left(1 + \frac{2\gamma_0 v + v^2}{\gamma_0^2 - z_i^2} \right)} &+ \sum_1^n t_i \frac{z_i \ln \left(\frac{1 + \frac{v}{\gamma_0 - z_i}}{1 + \frac{v}{\gamma_0 + z_i}} \right)}{(\gamma_0^2 - z_i^2) \left(1 + \frac{2\gamma_0 v + v^2}{\gamma_0^2 - z_i^2} \right)} \\ &= \sum_1^n t_i \frac{z_i \left\{ \ln \left(\frac{\gamma_0 + z_i}{\gamma_0 - z_i} \right) - 2\alpha_0 t_i \right\}}{(\gamma_0^2 - z_i^2) \left(1 + \frac{2\gamma_0 v + v^2}{\gamma_0^2 - z_i^2} \right)} \end{aligned}$$

Expanding the logarithms by Maclaurin's series and neglecting terms of the order of uv and v^2 in comparison with those of the order of u and v we have

$$\begin{aligned} u \sum_1^n t_i^2 + v \sum_1^n t_i \frac{t_i z_i}{\gamma_0^2 - z_i^2} &= \sum_1^n t_i \frac{1}{2} \left(\ln \frac{\gamma_0 + z_i}{\gamma_0 - z_i} - 2\alpha_0 t_i \right) \\ u \sum_1^n t_i \frac{t_i z_i}{\gamma_0^2 - z_i^2} + v \sum_1^n t_i \frac{z_i^2}{(\gamma_0^2 - z_i^2)^2} &= \sum_1^n t_i \frac{z_i}{(\gamma_0^2 - z_i^2)} \left(\ln \frac{\gamma_0 + z_i}{\gamma_0 - z_i} - 2\alpha_0 t_i \right) \end{aligned}$$

Expressing the natural logarithms in terms of common logarithms we have, putting

$$A_i \equiv t_i, \quad B_i \equiv \frac{z_i}{\gamma_0^2 - z_i^2}, \quad C_i \equiv \log \frac{\gamma_0 + z_i}{\gamma_0 - z_i} - M t_i,$$

$$K = \frac{\ln 10}{2} = 1.15129, \quad M = \frac{2\alpha_0}{\ln 10} = 0.86589\alpha_0;$$

$$\begin{aligned} u \sum_1^n t_i A_i^2 + v \sum_1^n t_i A_i B_i &= K \sum_1^n t_i A_i C_i \\ u \sum_1^n t_i B_i A_i + v \sum_1^n t_i B_i^2 &= K \sum_1^n t_i B_i C_i \end{aligned}$$

or in the usual notation of the method of least squares

$$u[AA] + v[AB] = K[AC]$$

$$u[BA] + v[BB] = K[BC]$$

From these equations u and v are readily computed, and hence the corrected values

$$\alpha = \alpha_0 + u$$

$$\gamma = \gamma_0 + v$$

are found.

6. *Application of the method of solution to other equations of reaction velocity.*

The well-known equation^a

$$\frac{dx}{dt} = k_1(1-x)^2 - k_2x^2$$

with the initial conditions

$$x=0 \text{ when } t=0$$

is reducible by the substitutions

$$\begin{aligned} z &= \frac{x}{1-x} & x &= \frac{z}{1+z} \\ \alpha &= \sqrt{k_1k_2} & k_1 &= \alpha\gamma \\ \gamma &= \sqrt{\frac{k_1}{k_2}} & k_2 &= \frac{\alpha}{\gamma} \end{aligned}$$

to the form we have considered, viz:

$$\frac{dz}{dt} = \frac{\alpha}{\gamma}(\gamma^2 - z^2) \quad (4)$$

with the initial conditions $z=0$ when $t=0$ its integral is, therefore,

$$\ln \frac{\gamma+z}{\gamma-z} = 2\alpha t \quad (6)$$

or

$$z = \gamma \tanh \alpha t \quad (7)$$

or

$$x = \frac{\gamma \tanh \alpha t}{1 + \gamma \tanh \alpha t} \quad (8)$$

^a Nernst, Walther, *Theoretische Chemie*, 5th edition, p. 564; 2d English edition, p. 568.

and the constants are determined by the equation

$$\ln \frac{\gamma + z_2}{\gamma - z_2} = \frac{t_2}{t_1} \ln \frac{\gamma + z_1}{\gamma - z_1} \quad (9)$$

if the criterion

$$\frac{z_2}{z_1} < \frac{t_2}{t_1} \quad (10)$$

is satisfied.

The more general equation ^a

$$\frac{dx}{dt} = k_1(a_1 - x)(b_1 - x) - k_2(a_2 + x)(b_2 + x)$$

with initial conditions $x=0$ when $t=0$, is reducible by a substitution of the form

$$x = \frac{\rho z - \sigma}{1 + z}$$

where ρ and σ are constants depending on a_1 , b_1 , a_2 , b_2 , but not on k_1 , k_2 , to the equation

$$\frac{dz}{dt} = \frac{\alpha}{\gamma}(\gamma^2 - z^2) \quad (4)$$

The initial conditions, however, are now not

$$t=0, \quad z=0 \quad (5)$$

but

$$t=0, \quad z = \frac{\sigma}{\rho} \equiv z_0$$

and hence the integral and the equation determining the constants are more complex.

The equation determining γ is

$$\ln \frac{\gamma + z_2}{\gamma - z_2} = \frac{t_2}{t_1} \ln \frac{\gamma + z_1}{\gamma - z_1} - \left[\left(\frac{t_2}{t_1} - 1 \right) \ln \frac{\gamma + z_0}{\gamma - z_0} \right]$$

and the criterion for the existence of a solution is

$$t_2 z_1 - t_1 z_2 - (t_2 - t_1) z_0 > 0$$

7. Calculation of x .

Using the data given in Tables 1 and 2 (pp. 11, 14), the values of γ and α , and hence of k_1 , k_2 , and K , have been calculated by the method of least squares as described above. The constants for charcoal are tabulated in Table 4.

^a Nernst, Walther, Theoretische Chemie, 5th edition, p. 543; 2d English edition, p. 542.

The last column of each of Tables 1 to 3 contains values of x calculated from the equation

$$x = \frac{4a}{a+1} \frac{\gamma \tanh \alpha t}{1 + \gamma \tanh \alpha t}. \quad (8)$$

The CO₂ used was nearly pure, and therefore $a=1$, whence,

$$x = \frac{2\gamma \tanh \alpha t}{1 + \gamma \tanh \alpha t}$$

The calculated and observed values of x agree within 2 or 3 per cent.

TABLE 4.—*Constants used in computation of x (charcoal).*

Tem- perature (°C.).	a ($a=1$)	γ ($a=1$)	k_1	k_2	k_1	$K = \frac{k_1}{k_2}$
800	0.0276	0.3568	0.03373	3.031	0.01908	0.006493
850	0.0612	0.5853	0.03443	3.238	0.07174	0.02216
900	0.09998	0.7711	0.02646	2.599	0.1540	0.05925
925	0.1297	0.8388	0.02291	2.298	0.2175	0.09465
1,000	0.3617	0.8853	0.04416	4.708	0.6404	0.1360
1,100	0.7921	0.9437	0.04588	5.275	1.495	0.2834

The curves in figures 3, 4, 5, and 6 were plotted from values of x (per cent CO) calculated from equation (8). The observed values are indicated by the small circles. By means of equation (8) it is possible to calculate the percentage of CO corresponding to any given gas velocity, providing α and γ or k_1 and k_2 are known. The values of k_1 and K , as well as α and γ , given in Table 4, exhibit a systematic variation with temperature. If an equation can be found which will express k_1 and K as a function of temperature, it will then be possible to calculate the percentage of CO for any time of contact and any desired temperature.

8. Variations of K and k_1 with temperature.

By thermodynamical considerations Van't Hoff has derived the following expressions for the variation of the constant of equilibrium, K , and the coefficient of reaction velocity, k , with temperature.

$$\frac{d(\ln K)}{d\theta} = -\frac{Q}{R\theta^2}$$

and

$$\frac{d(\ln k)}{d\theta} = \frac{A}{\theta^2} + B.$$

In these equations Q is the latent heat of reaction at the absolute temperature θ ; A is a function of Q , but is selected arbitrarily. B is

an arbitrary function of the temperature. By integration the latter equation becomes

$$\ln k = -\frac{A}{\theta} + B\theta + C, \quad 11$$

where C is an integration constant. The values of A , B , and C in equation (11) have been determined from the simultaneous values of k_1 and θ of Table 1. Table 5 contains the values of k_1 observed at various temperatures, k_1 (obs.), as well as the values of k_1 calculated from equation (11), k_1 (cal.). The agreement is remarkably good, showing that the velocity coefficient, k_1 , follows the differential equation which holds for most rates or reactions.

TABLE 5.—Variation of k_1 with temperature (charcoal).

$$[\ln k_1 = -\frac{50910}{\theta} - 0.0203\theta + 65.376]$$

Temperature (°C).	θ	k_1 (obs.).	k_1 (cal.).
800	1,073	0.020	0.021
850	1,123	0.073	0.064
900	1,173	0.154	0.159
925	1,198	0.217	0.237
1,000	1,273	0.640	0.629
1,100	1,373	1.49	1.53

In order to integrate the equation

$$\frac{d(\ln K)}{d\theta} = -\frac{Q}{R\theta^2}$$

it is first necessary to determine Q as a function of temperature. Kirchoff has shown that the increment of Q per degree centigrade is equal to the difference of the molecular heats of the factors and of the products of the reaction:

$$Q = Q_0 + (\sigma_{\text{co}_2} + \sigma_{\text{C}} - 2\sigma_{\text{CO}})\theta$$

Q_0 = The heat of reaction at $\theta = 0$.

σ = molecular heat at temperature θ .

σ_0 = molecular heat at temperature $\theta = 0$.

$$\sigma' = \frac{1}{2} \frac{d\sigma}{d\theta}$$

Assuming that σ is a linear function of temperature, which is very nearly true for gases

$$Q = Q_0 + \Sigma(\sigma_0)\theta + \Sigma(\sigma')\theta^2$$

$$\frac{d(\ln K)}{d\theta} = -\frac{1}{R} \left[\frac{Q_0}{\theta^2} + \frac{\Sigma(\sigma_0)}{\theta} + \Sigma(\sigma') \right]$$

$$\ln K = \frac{1}{R} \left[\frac{Q_0}{\theta} - \Sigma(\sigma_0) \ln \theta - \Sigma(\sigma')\theta + C \right] \quad (12)$$

The constants in this equation (12) may be evaluated by two different methods: by experimental determinations of Q , σ_0 , and σ' , or from four or more simultaneous observations of K and θ . The first method was adopted in this instance. The third column of Table 6 contains the values of K calculated by means of equation (12), taking

$$Q_0 = -40166$$

$$\Sigma(\sigma_0) = -2.055$$

$$\Sigma(\sigma') = .003104$$

$$C = 8.604$$

The values in the column marked K (obs.) are the constants obtained from the observed values of x and T in Table 1.

TABLE 6.—Values of K from experimental determinations.

$$[\ln K = -\frac{20235}{\theta} + 1.035 \ln \theta - 0.001564\theta + 8.604]$$

Temperature (°C.).	K (obs.).	K (cal.).	x_{∞} (obs.).	x_{∞} (cal.).	x_{∞} (obs.) Boudouard.
500	0.000007	0.021
600	0.00013	0.063
650	0.00046	0.185	0.39
700	0.00137	0.283
800	0.0065	0.0090	0.526	0.582	0.93
850	0.022	0.020	0.738	0.722
900	0.059	0.042	0.871	0.832
925	0.094	0.060	0.912	0.873	0.96
1,000	0.136	0.151	0.939	0.945
1,100	0.283	0.448	0.971	0.981
1,200	1.120	0.944
1,300	2.455	0.997
1,400	4.826	0.9985
1,500	8.671	0.9992
1,600	14.44	0.9996

In the fourth column of Table 6 are the observed values of x_{∞} , the amount of CO in equilibrium with CO₂ and charcoal at temperatures from 800° to 1,100° C., and in the fifth column the values of x corresponding to the values of K in the third column. For the specific heats of CO and CO₂, Langen's values were used, and for the specific heat of charcoal, Kunz's. Q_0 was calculated from the heats of combustion of carbon (amorphous) and carbon monoxide at ordinary temperatures.^a C was calculated from equation (12) using all the values of K (obs.) by the method of least squares.

The constants of equation (12) were calculated also by the method of least squares from simultaneous values of K (obs.) and θ , but the agreement was less satisfactory than in the former method.

The agreement between observed and calculated values of K and k_1 in Tables 5 and 6 shows that the increase of K or k_1 with temperature

follows Van't Hoff's laws. It is possible, therefore, by means of equations (11) and (12) and the values of the constants of these equations given in Tables 5 and 6 to compute K and k_1 for any desired temperature. For any temperature for which K and k_1 , and consequently k_2 ($k_2 = \frac{k_1}{K}$), have been computed, x , the per cent of CO corresponding to any time of contact, t , can then be calculated by means of equation (8).

TABLE 7.—Variation of x_{∞} (per cent CO in equilibrium with CO_2 and C) with temperature.

Temperature (°C.).	x_{∞} (cal.)	x_{∞} (obs.) Charcoal.	x_{∞} (obs.) Coke.	x_{∞} (obs.) Anthracite.
900	0.832	0.871	0.875
1,000	0.945	0.939	0.886
1,100	0.981	0.971	0.908	0.914
1,200	0.994	0.937	0.934
1,300	0.997	0.966	0.997

TABLE 8.—Variation of k_1 with temperature (coke).

$$[\ln k_1 = -\frac{47220}{\theta} - 0.009699\theta + 45.597.]$$

Temperature (°C.).	θ	k_1 (obs.).	k_1 (cal.).
900	1,173	0.0024	0.0023
1,000	1,273	0.021	0.023
1,100	1,373	0.121	0.134
1,200	1,473	0.473	0.410
1,300	1,573	1.38	1.48

TABLE 9.—Variation of k_1 with temperature (anthracite).

$$[\ln k_1 = \frac{31972}{\theta} + 0.02272\theta - 56.607.]$$

Temperature (°C.).	θ	k_1
1,100	1,373	0.119
1,200	1,473	0.237
1,300	1,573	0.579

The percentages of CO in equilibrium with CO_2 and charcoal, coke, and anthracite, respectively, x , are given in Table 7. The values of x in the second column of this table were calculated from the values of K in the third column of Table 6 by means of the equation

$$\frac{x^2}{1-x} = K \frac{R\theta}{P}$$

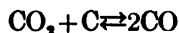
A comparison of figures 3, 5, and 6 shows that the reaction velocity is greatest with charcoal and lowest with anthracite. The tempera-

ture coefficient of k_1 was determined for anthracite and coke in the same manner as for charcoal. The observed and calculated values of k_1 are shown in Tables 8 and 9.

The constant, k_2 , in the equation

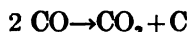
$$\frac{d(\text{CO})}{dt} = k_1(\text{CO}_2) - k_2(\text{CO})^2$$

is the coefficient of reaction velocity of the reaction



in the direction from right to left. At the temperatures of these experiments, $800^\circ - 1,300^\circ \text{C}$, the carbon produced by the decomposition of CO is in the form of lampblack, regardless of the form of carbon present in the reaction tube—charcoal, coke, or coal. At any one temperature, therefore, k_2 should be the same in all three cases. From a comparison of Tables 1 to 3 it will be seen that there is considerable deviation in the values of k_2 for the three forms of carbon used. This is doubtless due in part to experimental errors. There is a further consideration, however, to which attention should be called, viz, that the reaction in question is not strictly reversible.

The lampblack produced by the reverse reaction



is not identical, physically, with the form of carbon, charcoal, or coke, which is consumed in the formation of CO. Consequently, the law of chemical mass action is not strictly applicable. In the system under consideration, equilibrium would not be reached until all the carbon has been transformed to lampblack.

DISCUSSION AND CONCLUSIONS.

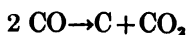
RESULTS OF BOUDOUARD'S INVESTIGATIONS.

The results of Boudouard's investigations of the percentage of CO in equilibrium with CO_2 and carbon, referred to on pages 6 and 7, are also given in the last column of Table 6. A comparison of Boudouard's values with those obtained in our experiments shows a decided disagreement, Boudouard's values for CO being higher in every case. The writers are unable to account for so great a discrepancy. It is to be noted, however, that in the analysis of his gases Boudouard apparently determined CO_2 directly and CO by differences, so that any air which might have been present in the gases would have been reported as CO.

It does not seem likely that errors in the measurement of temperature could be responsible for differences of so great a magnitude in the amount of CO formed.

As stated on page 9, and illustrated in figure 1, in all the experiments the hot junction of the thermocouple was imbedded in the carbon within the reaction tube, and the couple was calibrated at frequent intervals. Also, the cold junction was maintained at 0° C. The error in the temperature measurements could not exceed 15° or 20° C. Moving the couple through the portion of the tube occupied by carbon showed that there were no temperature differences in the carbon bed greater than 10° C. It is not apparent from Boudouard's account of his experiments what precautions were observed to insure accurate temperature measurements.

That the percentage of CO obtained in the writer's experiments was not reduced appreciably by the reverse reaction



taking place during the passage of the gases through the region of falling temperature at the exit end of the tube is indicated by the fact that no deposit of carbon could be observed at this portion of the tube.

REACTION KINETICS.

Van't Hoff has shown that the coefficient of reaction velocity, k , for a number of reactions increases between 2 and 2.5 fold for a rise of temperature of 10° C.

The reactions considered by Van't Hoff were investigated at temperatures below 100° C. At higher temperatures the rate of increase in k with rise of temperature is lower.

If for a given reaction, at a temperature of 100° C.

$$\frac{k_{\theta+10}}{k_{\theta}} = 2.5,$$

it follows from Van't Hoff's equation,

$$\frac{d(\ln k)}{d\theta} = \frac{A}{\theta^2},$$

that for $T=1,000^{\circ}$, $\theta=1,273^{\circ}$,

$$\frac{k_{\theta+10}}{k_{\theta}} = 1.08.$$

In table 10 the values of the ratio $\frac{k_{\theta+10}}{k_{\theta}}$ corresponding to the results in Tables 5, 8, and 9 indicate that the increase of k with rise in temperature is of the magnitude which might be expected for reaction velocities in homogeneous systems.

TABLE 10.—*Increase in k with a rise in temperature of 10° C.*

Form of carbon.	Temperature range ($^\circ$ C.)	$k_{\theta+10}$ k_{θ}
Charcoal	800-1,100	1.155
Coke	900-1,300	1.175
Anthracite	1,100-1,300	1.082

It has been previously stated (p. 20) that the speed of chemical reactions increases far more rapidly with rise of temperature than the rate of diffusion of gases.

Table 11 gives the observed increase in velocity of the reaction $\text{CO}_2 + \text{C} = 2\text{CO}$, and the increase in rate of diffusion over the same temperature intervals calculated on the assumption that diffusion increases with the square of the temperature.

TABLE 11.—*Relative rates of increase in reaction velocity and diffusion constant.*

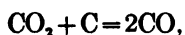
Form of carbon.	Temperature range ($^\circ$ C.).	Observed increase in reaction velocity (k_1).	Calculated increase in diffusion constant.
Charcoal	800-1,100	75 fold	1.64 fold
Coke	900-1,300	576 fold	1.80 fold
Anthracite	1,100-1,300	4.9 fold	1.31 fold

It seems probable from the high values of the increase of k_1 with rising temperature, found in our experiments and shown in the third column of this table, that the determining factor in speeds of the reactions under consideration is the velocity of chemical combination and not of diffusion.

But aside from the question as to whether or not the laws of reaction velocity apply to the heterogeneous reaction, $\text{CO}_2 + \text{C} = 2\text{CO}$, the equations given in this paper expressing x as a function of t , and k_1 and K as functions of the temperature, fit the experimental results obtained. These equations, therefore, furnish a means for calculating the percentage of CO formed for any value of t , the time of contact, and T , the temperature at which the reaction takes place.

SUMMARY.

1. The rate of formation of CO in the reaction



has been determined, with charcoal from 800° to $1,100^\circ$ C., with coke from 900° to $1,300^\circ$ C., and with anthracite from $1,100^\circ$ to $1,300^\circ$ C.

2. The differential equation for the velocity of the incomplete reaction, $\text{CO}_2 + \text{C} \rightleftharpoons 2\text{CO}$,

$$\frac{dx}{dt} = k_1 \left(a - \frac{a+1}{2} x \right) - k'_2 x^2$$

has been solved for k'_2 and k_1 , and it has been shown that the method is applicable to other cases.

3. Van't Hoff's laws for the variation of equilibrium constants and coefficients of reaction velocity with temperature have been applied to the values of k_1 and K obtained in these experiments and a close agreement found between observed and calculated values.

4. From a consideration of the temperature coefficients of k_1 , it is probable that velocity of chemical combination and not diffusion is the determining factor in the speed of the reaction.

5. By means of the equations expressing the laws referred to in paragraphs 2 and 3, it is possible to compute the percentage of CO, which is formed at any temperature and with any time of contact.

6. It has been shown that, for the production of a high percentage of CO, the producer fuel bed should have a temperature of $1,300^\circ \text{C}$. or over, and that increasing the depth of the hot portion of the bed will increase the percentage of CO generated and consequently the capacity of the producer, at first rapidly and then more and more slowly.

7. To minimize the production of CO in the boiler furnace the fuel bed should be thin. Increasing the velocity of the gas will tend to decrease rather than increase the percentage of CO formed.

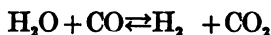
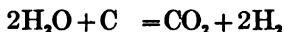
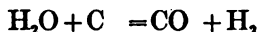
EFFECTIVE TEMPERATURES FOR WATER-GAS GENERATION.

By J. K. CLEMENT and L. H. ADAMS.

INTRODUCTION.

The object of this paper is to describe investigations which were carried on in the physical laboratory of the technologic branch of the United States Geological Survey to determine the influence of temperature on the rate of formation and on the composition of water gas, and to ascertain the most favorable conditions for the economical production of combustible gases from water vapor and carbon.

Incandescent carbon combines readily with water vapor, forming a mixture of hydrogen, carbon monoxide, carbon dioxide, and, as is shown by the results of the investigation described, a small quantity of methane. The principal reactions are represented by the equations



The composition of the gas obtained when a stream of superheated steam is passed through a bed of glowing carbon, as well as the quantity of steam decomposed in a given time, vary with the temperature, the rate of flow, and the kind of carbon used.

DESCRIPTION OF EXPERIMENTS.

APPARATUS AND METHODS.

The methods employed were substantially the same as had been used in the preceding investigation of the reactions between carbon dioxide and carbon (pp. 9, 10). Carbon—coke or charcoal—crushed and screened to pieces about 5 mm. in diameter was placed in a tube of refractory material (Berlin porcelain or quartz glass) and a current of superheated steam passed through it.

Heating coil and superheater.—The arrangement of apparatus is illustrated in figure 8. Steam was generated by means of an electric heating coil and passed through a spiral copper-tube superheater, and thence into the top of the reaction tube. By regulating the

current in the heating coil, the rate of flow of steam could be readily adjusted to any desired value, and its flow made uniform.

Electric furnace and reaction tube.—The electric furnace shown in figure 8 is of the same general type as that used in the experiments with carbon dioxide and carbon already described. The nickel heating coil of the earlier furnace, however, was replaced by one of "nichrome" wire obtained from the Driver-Harris Company, of Harrison, N. J. Compared with nickel, this alloy possesses the advantages of lower temperature coefficient of resistance, higher fusion point, and less tendency to become brittle at high temperatures.

A number of experiments were made with the furnace at a temperature of $1,300^{\circ}\text{C.}$, and it is not improbable that a somewhat higher temperature may be produced without destroying the heating coil.

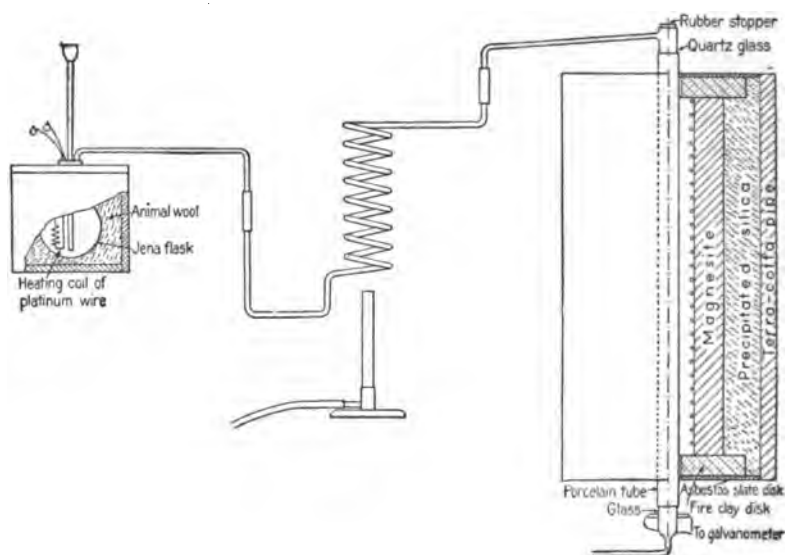


FIGURE 8.—Arrangement of heating coil, superheater and electric furnace.

Figure 8 shows the reaction tube in position in the furnace. By means of suitable rheostats any temperature up to $1,300^{\circ}\text{C.}$ could be readily obtained, and the temperature could be kept constant to within 1° or 2°C.

In order to have the temperature of the reacting material as nearly uniform as possible, only the central portion of the tube was filled with carbon. The lower part of the tube was filled with pieces of broken porcelain, which, in addition to forming a support for the carbon, reduced the size of the passageway, thereby increasing the velocity of the gas, through the region of variable temperature.

After three weeks' use at 800° and 900°C. —the furnace being allowed to cool at night during the first two weeks and being heated continuously during the third week—the quartz-glass tube was found to be completely devitrified. All experiments at $1,000^{\circ}\text{C.}$ and over and four experiments at 900°C. were made in a tube of Berlin porcelain.

Analysis of gases.—After passing through the reaction tube the gas was led in turn through an empty U-shaped drying tube and a tube containing calcium chloride, in which the moisture was condensed and weighed. The fixed gases were then collected over mercury and analyzed, in Babb's improved Orsat apparatus, for CO_2 , O_2 , CO , H_2 , and CH_4 . The analyses were checked from time to time by the Hempel method.

Measurement of temperatures.—Temperature measurements were made by means of a platinum-rhodium thermocouple and a high resistance millivoltmeter. At the higher temperatures the thermocouple wires deteriorated rapidly—probably because of the reducing action of the gases on the insulating tubes or on the ash of the coke—and the hot portion of the wires became brittle and fell to pieces. The couple had to be frequently repaired by removing the contaminated part and re-fusing the wires, and finally had to be replaced by a new couple.

TABLE 12.—Results of experiments with coke at 800° to 1,300° C.

No. of experiment.	Temperature (°C.).	Time of contact in seconds.	1 t	Composition of dry gas (per cent).					Composition of total gas (per cent).				
				CO_2 .	CO .	H_2 .	CH_4 .	Total.	H_2O .	CO_2 .	CO .	H_2 .	CH_4 .
1	2	3	4	5	6	7	8	9	10	11	12	13	14
1	800	1.019	0.983	4.8	34.4	42.5	-----	81.7	99.1	0.06	0.40	0.49	-----
2	800	.627	1.594	3.6	31.3	41.7	-----	76.6	99.4	.03	.25	.34	-----
3	800	.416	2.40	4.2	35.0	45.5	-----	84.7	99.6	.02	.16	.21	-----
4	900	8.35	.120	9.8	30.4	45.7	2.0	87.9	75.4	2.75	8.51	12.78	0.57
5	900	2.96	.337	6.8	39.8	47.5	1.9	96.0	89.5	.75	4.36	5.22	.21
6	900	2.88	.347	8.1	37.0	46.7	2.1	93.9	89.5	.91	4.16	5.25	.23
7	900	1.47	.679	7.9	36.0	46.6	1.8	92.3	92.9	.61	2.78	3.60	.14
8	900	1.37	.728	6.0	41.2	47.8	1.7	96.7	92.3	.48	3.29	3.82	.14
9	900	.721	1.387	6.4	40.1	47.9	2.2	96.6	95.0	.33	2.08	2.50	.11
10	900	.908	1.643	4.8	41.3	47.9	1.6	95.6	95.6	.27	2.34	2.71	.08
11	900	.800	2.000	6.0	40.8	47.5	1.9	96.2	96.2	.24	1.62	1.88	.08
12	900	.442	2.262	5.5	42.7	48.1	1.6	97.9	96.7	.19	1.45	1.63	.05
13	900	.428	2.336	5.4	41.3	47.4	1.8	95.9	97.4	.15	1.12	1.28	.06
14	900	.272	3.68	4.9	41.9	51.8	-----	98.6	98.0	.10	.86	1.06	-----
15	900	.245	4.08	5.9	39.5	48.0	1.2	94.6	97.9	.13	.90	1.09	.03
16	1,000	6.98	.142	13.6	28.6	49.3	2.6	94.1	69.8	4.38	9.16	15.80	.84
17	1,000	3.42	.292	10.7	33.6	50.3	2.0	96.6	78.4	2.40	7.53	11.28	.45
18	1,000	2.64	.379	9.6	35.1	48.4	1.9	95.0	81.3	1.89	6.92	9.56	.37
19	1,000	1.504	.666	8.7	37.3	49.7	1.9	97.6	84.2	1.41	6.05	8.07	.30
20	1,000	1.025	.976	7.8	38.0	48.5	1.6	95.9	88.7	.91	4.48	5.71	.19
21	1,000	.733	1.364	7.2	38.6	48.7	1.7	96.2	90.6	.70	3.76	4.75	.17
22	1,000	.437	2.29	8.0	38.2	49.3	1.8	97.3	93.7	.62	2.48	3.22	.11
23	1,000	.244	4.08	7.1	39.0	48.1	1.9	96.1	96.4	.27	1.47	1.81	.08
24	1,100	7.97	.1255	14.6	28.1	53.1	1.4	97.2	34.9	9.8	18.8	35.6	.90
25	1,100	1.970	.507	12.8	28.9	51.2	1.5	94.4	67.6	4.40	9.92	17.6	.51
26	1,100	1.034	.967	13.3	30.5	52.5	1.9	98.2	76.8	3.16	7.22	12.41	.44
27	1,100	.493	2.026	14.8	28.0	54.1	1.4	98.3	88.6	1.73	3.25	6.30	.16
28	1,100	.377	2.65	13.4	28.9	51.5	1.7	95.5	90.1	1.39	3.00	6.36	.18
29	1,100	.259	3.85	13.3	30.4	53.1	1.4	98.2	92.0	1.09	2.48	4.32	.11
30	1,200	11.06	.0903	.3	51.8	42.9	1.0	96.0	5.0	.3	51.3	42.5	1.0
31	1,200	4.48	.2224	.6	52.1	43.1	1.2	97.0	17.0	.5	44.6	37.0	.9
32	1,200	2.132	.468	.9	48.6	44.8	1.5	95.8	52.3	.4	24.2	22.3	.3
33	1,200	.866	1.155	7.4	39.3	49.4	1.2	97.3	74.8	1.92	10.18	12.80	.31
34	1,200	.478	2.064	11.5	33.1	53.4	1.1	99.1	80.8	2.23	6.43	10.37	.23
35	1,200	.337	2.96	3.6	46.3	47.0	1.9	98.8	83.0	.62	8.00	8.11	.32
36	1,300	4.32	.2314	.4	50.5	43.7	1.9	96.5	.0	.4	52.4	45.3	2.0
37	1,300	2.25	.443	.3	49.2	42.4	1.7	93.6	2.1	.3	51.5	44.3	1.8
38	1,300	1.633	.612	.3	49.5	43.9	1.8	95.5	7.7	.3	47.8	42.5	1.7
39	1,300	1.245	.802	.3	49.5	45.8	1.9	97.6	17.4	.3	41.9	38.8	1.6

The measurements of temperatures were accurate within about 5° below 1,100° C. and within 5° to 15° between 1,100° and 1,300° C.

RESULTS OF EXPERIMENTS.

EXPERIMENTS WITH COKE.

The results of the experiments with coke are given in Table 12. The time of contact, t , given in column 3, was calculated from the equation $t = S \frac{v \theta_r}{V \theta}$ where S is the time required to fill sample tube, θ and θ_r the absolute temperature of the carbon and of the sample tube, respectively, V the volume of the sample tube, and v the volume of the free passages through the carbon. The latter is determined by the dimensions of the portion of the reaction tube which is occupied by carbon and the weight and apparent density of the carbon. Since v can not be determined accurately and the formula for t rests on the assumption that the volume of the gas undergoes no change during the reaction, t is only approximately correct. The errors in t , however, are nearly constant, so that the values given in the table may be safely used for purposes of comparison.

TABLE 13.—Results of experiments with charcoal at 1,100° C.

No. of experiment.	Time of contact, in seconds, t .	$\frac{1}{t}$	Composition of gas (per cent).					
			H ₂ O.	CO ₂ .	CO.	H ₂ .	CH ₄ .	Total fixed gases.
101	6.92	0.1444	0.9	0.0	50.5	47.3	1.3	99.1
102	6.62	.1778	.9	.1	50.1	48.1	.8	99.1
103	3.37	.2968	12.3	.3	43.3	43.4	.7	87.7
104	1.773	.563	20.8	.4	39.6	39.0	.2	79.2

The reciprocal of the time of contact, $\frac{1}{t}$, is the velocity of the gas divided by the length of the carbon column or fuel bed. If the latter is taken as unity, then $\frac{1}{t}$ is the velocity. For example, if the depth of bed is 1 foot, then $\frac{1}{t}$ is the gas velocity in feet per second. The difference between 100 per cent and the values given in column 9 of the table is the amount of air present. In most cases the amount of oxygen found was one-fifth of this difference, indicating that the air leaked into the gas after the latter had left the reaction tube. When the per cent of water vapor in the gas leaving the tube is high, a slight trace of air in the wet gas may amount to several per cent of the water-free gas.

The hydrogen and methane content of the dry gas shows no systematic variation. It is probable that at the higher temperatures some hydrogen diffused through the wall of the porcelain tube, and that in the experiments at 1,200° and 1,300° C. the hydrogen content

is too low. At these temperatures the amount of hydrogen found is several per cent lower than would correspond to the relation $H_2 = CO + 2CO_2 - 2CH_4$.

Although the variation is not regular, it is apparent from the figures in columns 5 and 6 that with the rise in temperature the percentage of carbon dioxide in the dry gas decreases and that of carbon monoxide increases.

In the work of Harries, mentioned on pages 51 and 52, no methane is reported, and it is not apparent that a test was made for this gas.

EXPERIMENTS WITH CHARCOAL.

Table 13 contains the results of a series of experiments at $1,100^\circ C$. in which willow charcoal was used instead of coke. The results show that charcoal reacts with water vapor much more rapidly than coke. The water vapor was practically completely decomposed in 5 seconds.

DISCUSSION OF RESULTS.

Formation of methane.—The possibility of forming methane in the gas producer by the combination of steam and carbon probably has not been considered heretofore. The methane contained in producer gas is generally attributed to the destructive distillation of the fuel, and doubtless the greater portion of it is formed in this way. But the relatively large amounts of methane found in the experiments here described—usually about 2 per cent of the dry gas—make it highly probable that an appreciable part of the methane content of producer gas is due to the combination of steam and carbon. This conclusion is confirmed by a gas-producer test with coke, made in the fuel-testing plant of the United States Geological Survey, in which the gas contained an average of 0.2 per cent of methane.

As to the possibility of the methane found in these experiments being due to impurities in the coke, it was repeatedly proven that heating the charge for several hours produced no reduction in the amount of methane obtained when water vapor was passed over the coke. Previous to beginning the experiments with charcoal the charge was heated for 2 hours at $1,100^\circ C$. in order to drive off the volatile content of the charcoal. In this connection recent experiments by Mayer, Henseling, and v. Altmeyer^a are of interest.

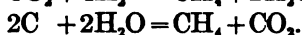
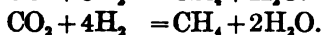
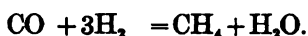
By passing a mixture of carbon dioxide or carbon monoxide and hydrogen over a catalyser of finely divided nickel or cobalt, heated to 200° to $500^\circ C$., these investigators have obtained large quantities of methane—50 to 60 per cent.

Under favorable conditions the carbon monoxide could be completely reduced to methane. By passing hydrogen over carbon which

^aJ. für Gasbel, vol. 52, 1909, pp. 166, 194, 238, 326.

had previously been deposited on the nickel catalyser a gas containing over 90 per cent CH_4 was obtained at a temperature of 245°C . A mixture of carbon monoxide and water vapor passed over a nickel catalyser at 300°C . yielded 18.8 per cent methane. Finally nitrogen saturated with water vapor at 80° to 90°C . was led over finely divided carbon deposited on nickel, at 246° , 286° , and 330°C ., and as much as 9.18 per cent of methane was found in the resultant gas.

According to Mayer and v. Altmeyer, methane is 98.5 per cent dissociated at 850°C . and 99.5 per cent at $1,000^\circ \text{C}$. At $1,200^\circ \text{C}$. the dissociation of methane is practically complete. The amounts of methane shown in column 14, Table 13, at $1,000^\circ \text{C}$. and above are too high to be in equilibrium with hydrogen, and therefore can not be entirely accounted for by the reaction, $\text{C} + 2 \text{H}_2 = \text{CH}_4$. The equilibrium between methane, hydrogen, and carbon, however, might be brought about by one or more of the following reactions:



While at temperatures between 200° and 500°C . these reactions take place only under the influence of a catalyser, it is not improbable that at higher temperatures they may take place without a catalyser.

RELATION OF YIELD AND COMPOSITION OF GAS TO TEMPERATURE AND TIME OF CONTACT.

DISCUSSION OF RESULTS.

Figures 9 to 16 show the variation of the amount of water vapor decomposed and of the composition of the gas with temperature and gas velocity. In every case the ordinate represents the percentage composition by volume of the gas. In figures 11 and 16 the abscissa is the temperature of the reaction in degrees centigrade. In the other figures the abscissa represents the reciprocal of the time of contact, $\frac{1}{t}$, which for constant dimensions of fuel bed is proportional to the gas velocity: $\frac{1}{t} = \frac{\text{velocity of gas}}{\text{depth of bed}} = \frac{v}{l}$. If $l = 1$ foot, $v = \frac{1}{t}$ = velocity in feet per second.

In figure 9 is shown the per cent of fixed gas (100 – the per cent of water vapor) formed at 900° to $1,300^\circ \text{C}$. The results for 800°C . are not shown. The amount of water decomposed at this temperature was always less than 1 per cent. The ordinates for zero velocity ($t = \infty$) were taken from results of experiments by Harries which will be discussed later. (See pp. 51, 52).

It will be seen that below 1,200° C., even with the lowest velocities reached—about one bubble of gas per second—the per cent of fixed gas formed is much lower than that found by Harries. With

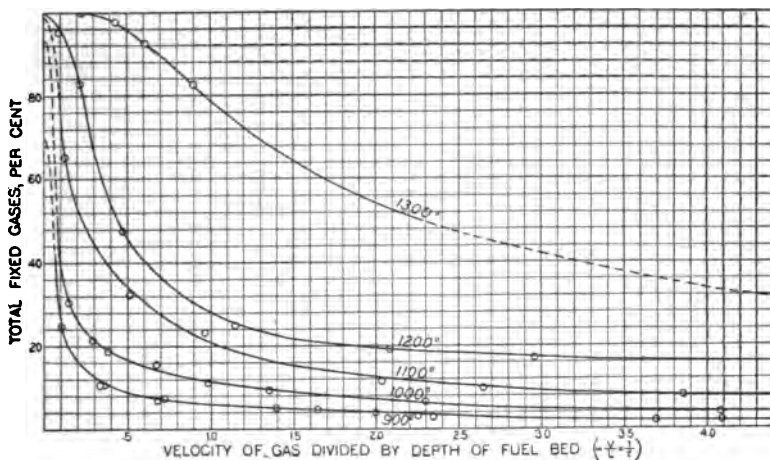


FIGURE 9.—Variation of percentage of total fixed gases formed with change of gas velocity.

increase in gas velocity the per cent of water decomposed falls off very rapidly at first, until a velocity of about 1 foot per second (assuming depth of fuel bed = 1 foot) is reached. Beyond this point the decrease of the per cent of fixed gas formed with increasing velocity

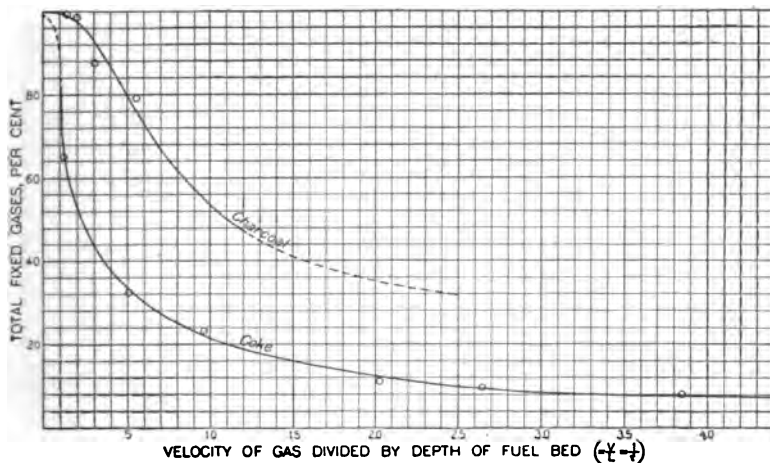


FIGURE 10.—Variation of percentage of total fixed gases formed from coke and from charcoal, at a fuel-bed temperature of 1,100° C., with change of gas velocity.

is slow. Below 1,200° C., with velocities of the magnitude found in producer operation (over 1 foot per second), less than one-fifth of the water vapor passed over the carbon is decomposed.

The results of the experiments with charcoal at 1,100° C. are represented by the upper curve of figure 10. The lower curve represents the results with coke at the same temperature. A com-

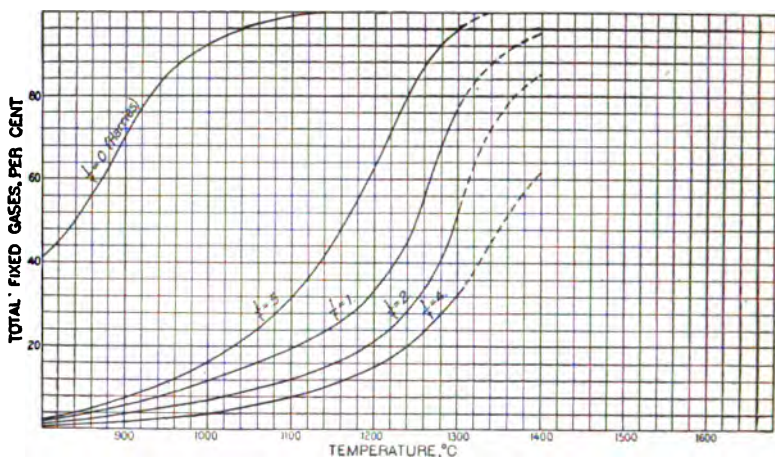


FIGURE 11.—Variation of percentage of total fixed gases formed with change of fuel-bed temperature.

parison of the two curves shows that, except with very low velocities, more than double the percentage of fixed gas is obtained with charcoal than with coke.

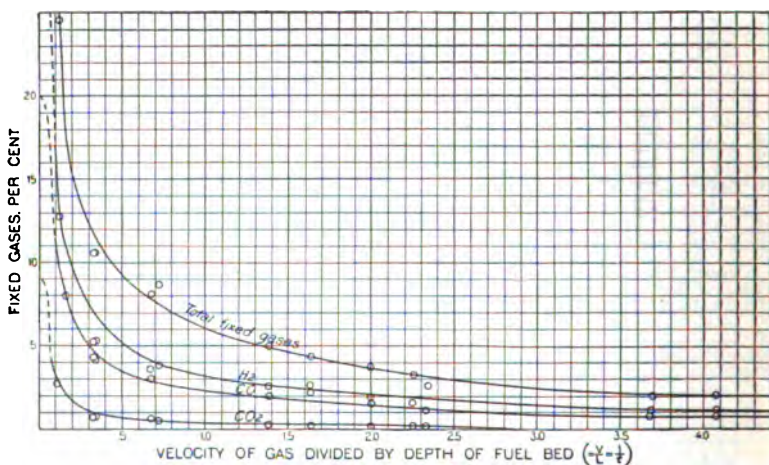


FIGURE 12.—Variation of percentage of fixed gases formed at a fuel-bed temperature of 900° C., with change of gas velocity.

The influence of temperature on the amount of water decomposed is further illustrated by figure 11, in which the percentage of fixed gas formed from coke and water vapor at various gas velocities is plotted as

a function of the temperature. Each curve corresponds to a particular velocity or time of contact. The upper curve, $\frac{1}{t}=0$, was plotted from Harries's values and represents the condition of equilibrium. It shows

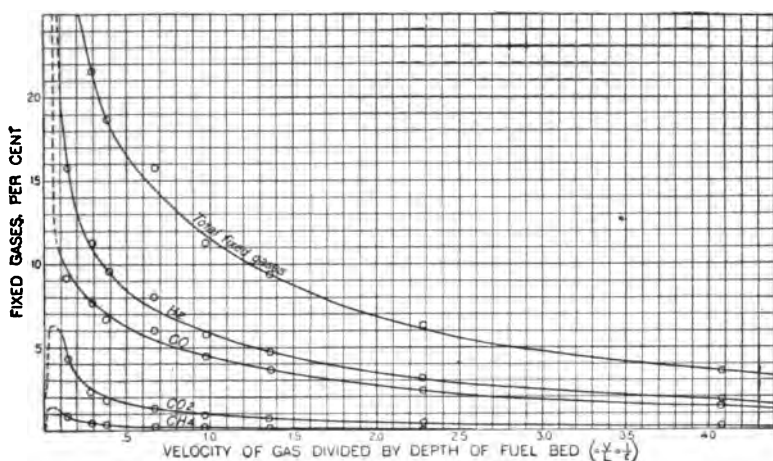


FIGURE 13.—Variation of percentage of fixed gases formed at a fuel-bed temperature of 1,000° C., with change of gas velocity.

that with infinite time of contact, or zero velocity, a temperature of 1,100° C. will be required in order for the decomposition of the water vapor to be practically complete. With a velocity of 0.5 foot per second

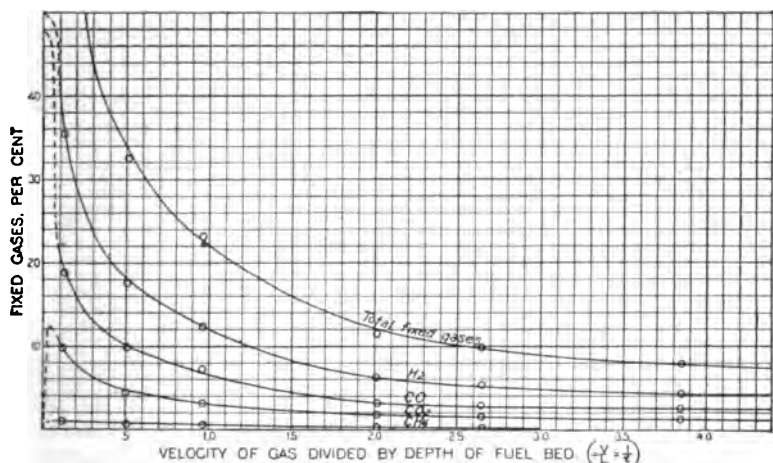


FIGURE 14.—Variation of percentage of fixed gases formed at a fuel-bed temperature of 1,100° C., with change of gas velocity.

(depth of fuel bed = 1 foot) and a time of contact of 2 seconds, the decomposition of the water vapor will be nearly complete at 1,300° C. To obtain the same amount of fixed gases with a velocity of 2 feet

per second and the same depth of fuel bed, the temperature must be raised at least 100°C . But, since $\frac{v}{l} = \frac{1}{t}$, if v and l are increased in

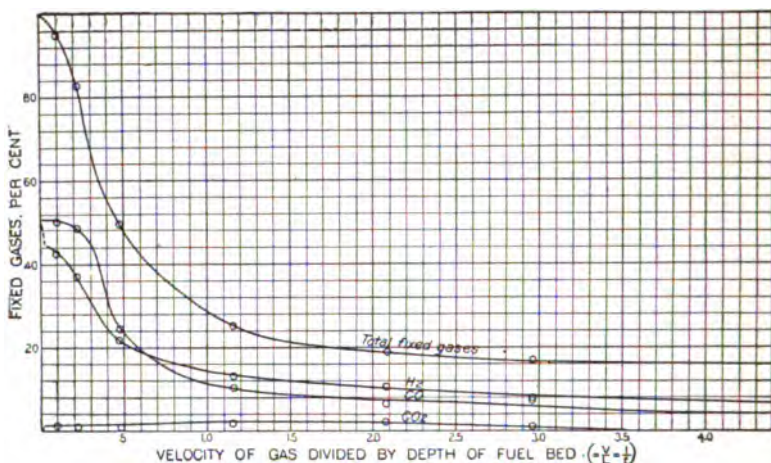


FIGURE 15.—Variation of percentage of fixed gases formed at a fuel-bed temperature of $1,200^{\circ}\text{C}$, with change of gas velocity.

the same proportion, and $\frac{1}{t}$ remains constant, the amount of fixed gases formed will not change as long as the temperature remains

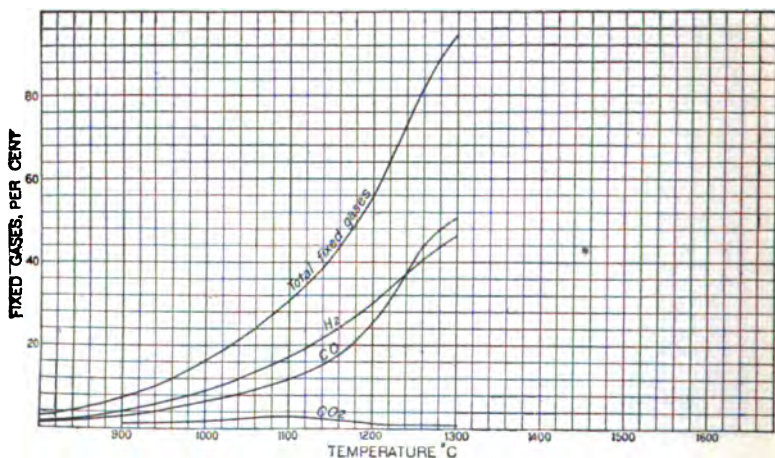


FIGURE 16.—Variation in composition of gas with rise of temperature, when the time of contact is 2 seconds ($\frac{1}{t} = 0.5$).

the same. Thus, with a temperature of $1,300^{\circ}\text{C}$ the percentage of steam decomposed in a fuel bed 10 feet in depth with a gas velocity of 10 feet per second will be the same as in a fuel bed 1 foot deep

with a gas velocity of 1 foot per second. In both cases 77 per cent of the steam will be decomposed.

The variation of the composition of the gas with velocity at temperatures of 900° to 1,200° C. is shown graphically in figures 12 to 15. The curves through the observed points are continued by dotted lines to the values found by Harries, which are assumed to correspond to zero velocity. The curves for H_2 and CO have the same general direction as the curve for total fixed gases. The curve for H_2 is higher than that for CO by an amount which agrees with the relation $H_2 = CO + 2 CO_2 - 2 CH_4$, except at 1,200° C. At this temperature the hydrogen content is low—especially at the lower velocities—and it is probable that some of the hydrogen diffused through the walls of the porcelain tube. As the rate of diffusion through the wall would be independent of the gas velocity inside of the tube, its effect on the composition of the gas would be more marked at low velocities.

The variation of the various components of the gas with temperature and with constant gas velocity, or time of contact, is shown in figure 16. It will be seen that the combustible constituents, CO and H_2 , increase steadily with rising temperature until a temperature is reached—slightly over 1,300° C.—at which the decomposition of the steam is practically complete. The CO_2 curve, on the other hand, reaches a maximum at about 1,100° C., and then gradually falls off with rising temperature. The curves in figure 16 correspond to a value of $\frac{1}{t} = 0.5$, $t = 2$ seconds. With a higher velocity and lower time of contact, the amount of fixed gases obtained at any given temperature would be less, and a correspondingly higher temperature would be required for complete decomposition of the steam.

EQUILIBRIUM BETWEEN CARBON DIOXIDE, CARBON MONOXIDE, AND CARBON.

At temperatures of 900° to 1,100° C., except with very low velocities, the per cent of CO_2 present is roughly proportional to the amount of water vapor decomposed. The lower the velocity and the greater the time of contact, the greater is the amount of CO_2 obtained. The results presented in the preceding paper, as well as the investigation of Boudouard ^a and of Mayer and Jacoby, ^b show that at temperatures of 1,000° C. and above, CO_2 combines readily with carbon to form CO , as expressed by the equation $C + CO_2 = 2 CO$, and that above 1,100° C. this transformation is

^a Boudouard, *Ann. Chem. Phys.*, vol. 24, 1901, p. 1.

^b Mayer and Jacoby, *J. für Gasbel.*, vol. 52, 1909, pp. 282, 305.

practically complete. According to Table 2 (p. 14), a mixture of CO and CO₂ in equilibrium with coke, and under a pressure of 1 atmosphere, will contain the gases in the following proportions:

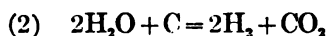
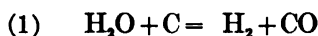
Composition of a mixture of CO and CO₂ when in equilibrium with coke.

Temperature (° C.).	CO (per cent.).	CO ₂ (per cent.).
900	83.2	16.8
1,000	94.5	5.5
1,100	98.1	1.9
1,200	99.4	.6
1,300	99.7	.3

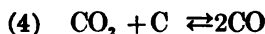
At any given temperature, in a mixture of CO and CO₂ in equilibrium with carbon, $\frac{(\text{CO})^2}{\text{CO}_2} = \text{constant}$. In mixtures containing about 50 per cent CO, therefore, the CO₂ content will be even less than the values given in the table. As the gas velocity approaches zero, then, the curves showing the percentage of CO₂ will have to pass through a maximum. The CO₂ curve for 1,200° C., figure 15, has a maximum at about $\frac{1}{t} = 2$. If the CO₂ curves for 1,000° and 1,100° C. are continued to $\frac{1}{t} = 0$, they will have to pass through a maximum and then fall off toward the percentages of CO₂ corresponding to the $\frac{(\text{CO})^2}{\text{CO}_2}$ ratios in the above table. The dotted lines in figures 13 and 14 show the probable direction of the CO₂ curves.

THE WATER-GAS REACTIONS.

The process of the formation of water gas is by no means a simple one. Hydrogen, carbon monoxide and carbon dioxide may be formed primarily by the reactions:



These products may undergo a readjustment in accordance with the equations:



As reactions (1) and (2) can not be studied separately, it is difficult to determine the relative importance of the individual reactions in

the formation of water gas. A consideration of the equilibrium constants of reactions (3) and (4) indicates the direction in which these reactions may take place: The ratio $\frac{H_2O \times CO}{H_2 \times CO_2}$ calculated from the results in Table 12 is much greater (usually ten times) than the constants determined by Luggin, Hahn, and Haber.^a Reaction (3) therefore, according to the law of chemical mass action, can take place only from left to right; that is, in the direction in which H_2 and CO_2 are formed. Similarly, since the per cent of CO_2 present at temperatures above $1,000^\circ C$ is in excess of the amount which could be in equilibrium with carbon and CO , reaction (4) can take place only in the direction in which CO is formed. Carbon dioxide may be formed by reactions (2) and (3) but not by reaction (4). The maximum in the CO_2 curves for $1,000^\circ$, $1,100^\circ$, and $1,200^\circ C$., shown in figures 12 to 14, indicates that the CO_2 formed by reactions (2) and (3) is afterwards broken up by reaction (4).

The previous investigation of the velocity of the reaction (4) yielded values too low to account for the rapid formation of CO in the experiments with water vapor and carbon. The greater part of the CO found, then, must be due to reaction (1), and this reaction seems, therefore, to be the dominant one. There is not sufficient evidence to demonstrate whether the increase in the ratio $\frac{CO}{CO_2}$, with rise in temperature, is due to the increase in the speed of reaction (1) over reaction (2), or to the increase in the velocity of reaction (4).

RESULTS OF PREVIOUS INVESTIGATIONS.

The action of water vapor on carbon at high temperatures has been investigated by Harries^b and Farup.^c The latter conducted experiments at temperatures between 821° and $911^\circ C$. He passed a stream of nitrogen and water vapor through a porcelain tube containing a carbon rod 4 mm. in diameter and about 10 cm. long and determined the amount of water decomposed. The greatest change in water-vapor concentration was from 44.3 to 35.9 per cent by volume. The gas velocity was quite low, the average time of contact being about 5 minutes.

Farup's results are of interest in illustrating the extreme sluggishness of the reactions between carbon and water vapor at temperatures below $900^\circ C$.

^a See p. 52.

^b J. fur Gasbel., 1894, p. 82; also Haber, Thermodynamics of technical gas reactions, London, 1908, p. 138.

^c Zeit. f. Anorgan. Chem., vol. 50, 1906, p. 276.

Harries passed water vapor over charcoal and determined the composition of the resulting gas. His results are shown in the following table:

TABLE 14.—*Harries's experiments with water vapor and charcoal.*

Temperature. (°C.).	Gas velocity per second (liters).	H ₂ .	CO.	CO ₂ .	H ₂ O.	$K = \frac{(H_2O)(CO)}{(CO_2) H_2}$	K (cal.) (Luggin).
674	0.9	8.41	0.63	3.84	87.12	1.70	0.49
758	1.8	22.28	2.67	9.23	65.82	.85	.70
838	3.28	32.77	7.96	12.11	47.15	.94	.98
861	5.3	36.48	11.01	13.33	39.18	.89	1.07
954	6.3	44.43	32.70	5.66	17.21	2.25	1.41
1,010	6.15	47.30	48.20	1.45	3.02	2.12	1.65
1,060	9.8	48.84	46.31	1.25	3.68	2.78	1.88
1,125	11.3	50.73	48.34	.60	.303	.48	2.11

As he does not state the capacity of the containing vessel and the quantity of charcoal used, the time of contact can not be determined. It is not apparent, therefore, whether the time of contact was sufficiently great for equilibrium to be established, although Luggin has used the results to compute the equilibrium constant,

$$K = \frac{C_{H_2O} \times C_{CO}}{C_{CO_2} \times C_{H_2}}$$

of the reaction $H_2O + CO \rightleftharpoons CO_2 + H_2$. In the expression for K , C_{H_2O} , C_{CO} , C_{CO_2} , and C_{H_2} denote the partial pressures of the reacting gases. The values of K given in the last column of Table 14 were calculated by means of the equation,

$$\log K = -\frac{2232}{\theta} - 0.08463 \log \theta - 0.0002203\theta + 2.4943.$$

Here θ is the absolute temperature.

From the results of more recent investigations of the water gas equilibrium by Hahn,^a and with the aid of Schreber's values for the gases taking part in the reaction, Haber has obtained the following expression for K ,

$$\log K = -\frac{2116}{\theta} + 0.783 \log \theta - 0.00043 \times \theta$$

The fairly good agreement between the values of K computed from Harries's results and those obtained by Luggin, Hahn, and Haber indicates that the various constituents in the gases obtained by Harries were in equilibrium with one another, though not necessarily with the solid carbon.

^a Zett. f. phys. Chem., vol. 44, p. 510, and vol. 48, p. 735.

Harries's results, then, give us the composition of the gas formed when steam is passed over incandescent carbon with very low velocities. They have been used, therefore, to plot the upper curve in figure 11, and in determining the probable intersection with the vertical axis of the curves in figures 9 and 12 to 14.

APPLICATION OF RESULTS TO GAS-PRODUCER PRACTICE.

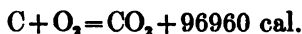
ADVANTAGES OF HIGH TEMPERATURE AND DEEP FUEL BED.

The previous investigation of the formation of carbon monoxide from carbon dioxide and carbon led to the conclusion that, in order that the reaction $\text{CO}_2 + \text{C} = 2\text{CO}$ in the fuel bed of the gas producer may be nearly complete and take place in the shortest possible time, a high temperature, not less than $1,300^\circ \text{C}$., must be maintained. The higher the temperature the more rapid will be the formation of carbon monoxide; also, since the amount of carbon monoxide formed increases with the time the gases are in contact with incandescent carbon, the greater the depth of the incandescent portion of the bed the greater will be the percentage of carbon monoxide obtained.

The present investigation leads to substantially the same conclusions with regard to the formation of hydrogen and carbon monoxide from steam and carbon. With zero velocity (time of contact $= \infty$) the decomposition of water vapor will be practically complete at $1,100^\circ \text{C}$. With a steam velocity of 0.5 foot per second (depth of bed = 1 foot) a temperature of $1,300^\circ \text{C}$., and with a gas velocity of 1 foot per second a temperature of over $1,400^\circ \text{C}$. will be required for complete decomposition of water vapor. (See fig. 11.) If

$\frac{1}{t} = 4.0$ (time of contact $= \frac{1}{4}$ second) 8 per cent of water vapor will be decomposed at $1,100^\circ \text{C}$., 15 per cent at $1,200^\circ \text{C}$., 32 per cent at $1,300^\circ \text{C}$., and about 60 per cent at $1,400^\circ \text{C}$. An increase of 100° between $1,100^\circ$ and $1,400^\circ \text{C}$. doubles the percentage of water vapor decomposed with a given gas velocity. The same percentage of water vapor may be decomposed at $1,300^\circ \text{C}$. with a gas velocity of 2 feet per second, as at $1,170^\circ \text{C}$. with a gas velocity of 0.5 foot per second. The percentage of water vapor decomposed by incandescent coke depends on two factors, the temperature and the time of contact, t . Since $\frac{1}{t} = \frac{\text{velocity of gas}}{\text{depth of bed}} = \frac{v}{l}$ at a given temperature, the same percentage of water may be decomposed with a high velocity and a deep bed of incandescent coke as with a low velocity and a shallow bed, providing the ratio $\frac{v}{l}$ is the same in each case. High capacity and high percentage of fixed gases formed require, then, high temperature and deep fuel bed.

Of the various chemical reactions which take place in the fuel bed of the gas producer one only is exothermic—that is, accompanied by an evolution of heat:

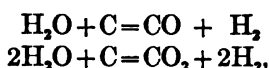


All other transformations, the distillation of hydrocarbons, the reduction of CO_2 to CO , the reactions between water vapor and carbon are endothermic—accompanied by an absorption of heat.

EFFECT OF STEAM.

The highest temperature in the fuel bed, and consequently the most rapid production of CO and the lowest amount of CO_2 , will be obtained when dry air is supplied to the gas producer. The introduction of steam will reduce the temperature of the bed, and thereby retard the formation of CO from CO_2 , and C and lower the capacity of the producer. Using an air blast alone Wendt^a obtained a gas containing 31 per cent CO and less than 1 per cent CO_2 , while with air and steam in the same producer he obtained from 19.2 to 28.6 per cent CO and from 5.5 to 10.5 per cent CO_2 . In the former case the temperature of the fuel bed was $1,400^\circ \text{C}$. and in the latter the highest temperature recorded was $1,110^\circ \text{C}$.

The object of adding steam to the blast of the gas producer is threefold—by reducing the temperature of the fuel bed to avoid or hinder the fusion of the ash and the formation of clinker; to prevent the destruction of the furnace walls from fusion or slagging; and to reduce the heat losses in the gases leaving the producer. When steam is introduced with the air, part of the energy which would otherwise leave the producer in the form of heat of the gases is absorbed by the endothermic reactions,



and thus made available for doing work in the gas engine or for heating purposes.

On account of the high percentage of CO_2 formed when relatively large quantities of steam are added it is doubtful whether a real gain in efficiency is accomplished. In fact, the results of investigations by Bone and Wheeler^b on the use of steam in gas producers show that with increasing amounts of steam there is a gradual decrease in efficiency as well as in the quality of the gas. Their tests were made

^a Wendt, Karl, *Untersuchungen an Gaserzeugern*; Stahl und Eisen, 1906, vol. 26, p. 1184.

^b Bone, W. A., and Wheeler, R. W., *Use of steam in gas producers*, J. Iron and Steel Inst., 1907, vol. 73, p. 126.

in a Mond producer, with a rated capacity of 1,600 pounds of coal per hour. Some of the results are given in the following table:

Gas-producer tests by W. A. Bone and R. W. Wheeler.

Water per pound coal fired.	Air per pound coal fired.	CO ₂ .	CO.	H ₂ .	CH ₄ .	Total combustible gas.	Efficiency.
<i>Pounds.</i>	<i>Cu. ft.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	
0.45	36.90	5.25	27.30	16.60	3.35	47.20	0.715
.55	34.90	6.95	25.40	18.30	3.40	47.10	.687
.80	36.80	9.15	21.70	16.95	3.40	44.70	.660
1.10	36.90	11.65	18.35	21.80	3.35	43.50	.640
1.55	37.10	13.25	16.05	22.60	3.50	42.20	.604

Bone and Wheeler suggest as a possible explanation of the increased amount of CO₂ obtained, when a great amount of steam is used, the preponderance of the reaction $2\text{H}_2\text{O} + \text{C} = \text{CO}_2 + 2\text{H}_2$ at low temperatures. In view of the low velocity of the reaction $\text{CO}_2 + \text{C} = 2\text{CO}$ at low temperatures, as shown by the results of the previous investigations (pp. 7-15), it is probable that the high CO₂ content of producer gas obtained when the temperature of the fuel bed is low is due largely to the retardation of the latter reaction.

A further disadvantage resulting from the use of large quantities of steam in the producer is indicated by the curves in figures 9 and 11. On account of the cooling of the fuel bed only a small part of the steam combines with the carbon to form combustible gases. The remaining undecomposed portion passes out of the producer, carrying with it a large amount of heat. In case the steam is generated in an auxiliary boiler, the fuel required for its production will materially reduce the efficiency of the plant.

FORMATION OF CLINKER.

The tendency of the different types of coal to form clinker depends on the quantity and character of the ash. With coals high in ash and when the ash is readily fusible, in the types of producer now in use, the introduction of a fairly large amount of steam is necessary to avoid the formation of large clinkers. The enormous increase in the rate of gasification with rise in temperature, as shown in the experiments described, suggests, however, the possibility of operating a producer at very high temperatures—1,500° C. or more—by providing for the removal of the slag in a liquid state after the manner of blast-furnace practice.

By using a hot blast it should be possible to maintain a temperature of 1,500° or 1,600° C. in the fuel bed. An extrapolation of the curves in figure 11 and in figure 7 of the preceding chapter indicates

that at these temperatures a capacity many times greater than that of current practice could be realized. A producer embodying this principle would be adapted to coals containing readily fusible ash, and might even make available certain coals which, because they clinker badly, have no market at present.

CONCLUSION.

The results presented show that a high rate of gasification combined with a high percentage of carbon monoxide and a low percentage of carbon dioxide and water vapor requires a high temperature in the fuel bed.

The higher the temperature the better will be the quality of the gas and the greater the capacity of the producer.

The use of large amounts of steam is inconsistent with the realization of high temperature, and is, therefore, to be avoided.

APPLICATION OF RESULTS TO WATER-GAS GENERATION.

In the process of making water gas the generator is supplied alternately with air and steam. During the period of blowing with air the temperature of the bed is raised as a result of the heat liberated by combustion. When the air blast is shut off and steam is blown through the incandescent fuel bed, heat is absorbed partly in raising the temperature of the steam to that of the fuel bed and partly by the endothermic reactions between steam and carbon. Consequently the fuel bed is rapidly cooled. The average temperature of the bed during the period in which water gas is being made depends on the duration of the periods and on the rate of supply of steam and air. The greater the air supply and the less the steam supply the higher will be the average temperature.

Writers on the subject of the water-gas processes are not entirely in agreement as to the temperature at which the generator can be most economically operated. On account of the loss of heat through the partial reduction of CO_2 to CO at high temperatures during the hot blast, Jüpner von Jonstorff^a concludes that it is "advantageous not to raise the temperature of the producer above 900°C ." On the other hand, Rollin Norris^b writes: "The hotter the fire, the greater the quantity of steam it can decompose per minute." The fire should be as hot as possible at the beginning of the run and the run should be cut off before the rate of decomposition is too low. In experiments made by G. W. McKee,^c in which steam was blown through a producer for 8 minutes at a uniform rate, it was found that the rate of formation of gas fell off rapidly between the third and fifth

^a Jüpner von Jonstorff, *Hans, Water gas, its treatment and properties*; translated by Oskar Nagel, *Electrochemical and Metal Industry*, 1909, p. 19.

^b Norris, Rollin, *Notes on capacities of water-gas sets*; *Am. Gas Light Jour.*, vol. 85, 1906, p. 46.

^c McKee, G. W., *Jour. Soc. Chem. Ind.*, vol. 22, 1903, p. 1325.

minutes of the run. By reducing the steam supply one-half after the fourth minute a considerable saving in coke was effected and the CO_2 in the gas was lowered.

As a result of experiments made at the Laclede plants in St. Louis, D. G. Fisher^a recommends that the steam supply be reduced or cut off at the end of the fourth minute.

Although the investigation described in the preceding pages was undertaken primarily to determine the conditions governing the formation of producer gas, the results have an important bearing on the water-gas process. They show that although with very low rates of steam supply the decomposition of the steam may be complete at $1,100^\circ \text{C}$., with higher rates of steam supply, such as are desirable in practice, a much higher temperature, $1,300^\circ$ or $1,400^\circ \text{C}$., is required to obtain complete decomposition. The highest efficiency will be obtained by raising the temperature of the bed during the blast as high as is possible without injury to the producer. As the bed cools during the run with steam, the steam supply should be gradually reduced, and when the temperature has dropped to $1,000^\circ \text{C}$. the steam should be cut off.

PUBLICATIONS ON FUEL TESTING.

The following publications, except those to which a price is affixed, can be obtained free by applying to the Director of the Bureau of Mines, Washington, D. C. The priced publications can be purchased from the Superintendent of Documents, Government Printing Office, Washington, D. C.

PUBLICATIONS OF THE UNITED STATES GEOLOGICAL SURVEY.

BULLETIN 261. Preliminary report on the operations of the coal-testing plant of the United States Geological Survey at the Louisiana Purchase Exposition, in St. Louis, Mo., 1904; E. W. Parker, J. A. Holmes, M. R. Campbell, committee in charge. 1905. 172 pp. 10 cents.

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^a Fisher, D. G., water-gas practice; Am. Gas Light Jour., vol. 85, 1906, pp. 139, 178.

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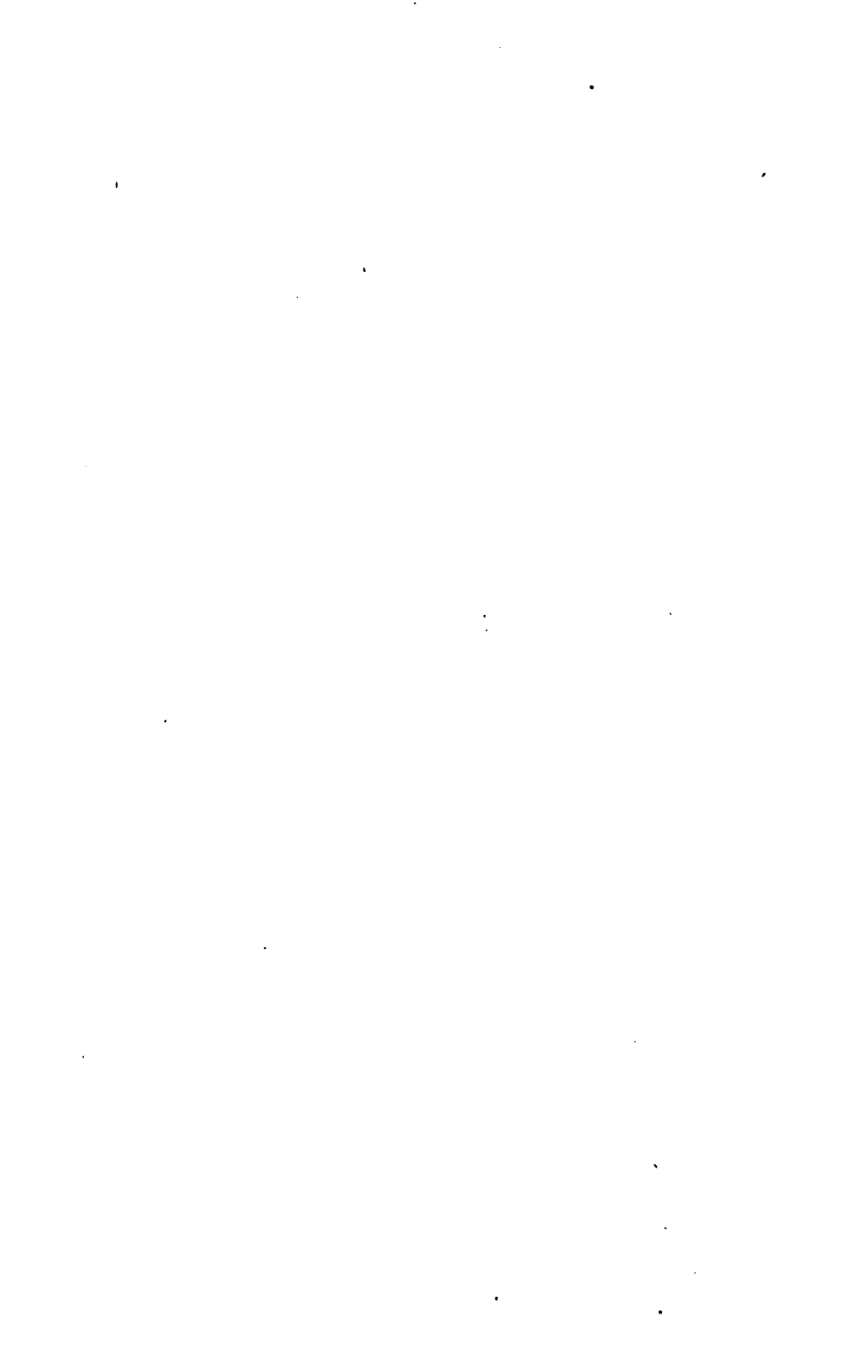
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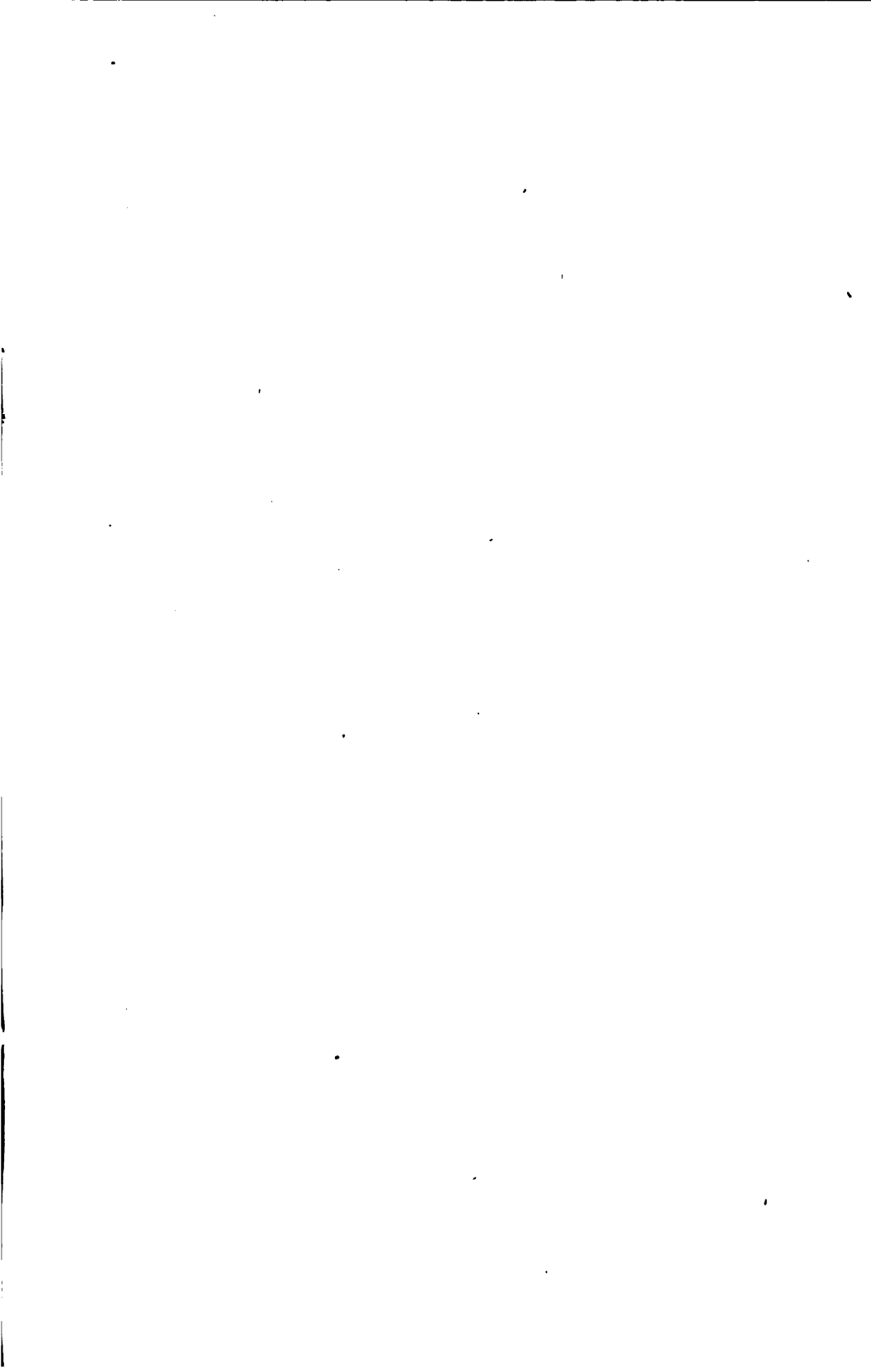
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Bulletin 8

DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

JOSEPH A. HOLMES, Director

THE FLOW OF HEAT
THROUGH FURNACE WALLS

BY

WALTER T. RAY AND HENRY KREISINGER



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THE FLOW OF HEAT THROUGH FURNACE WALLS.

By WALTER T. RAY AND HENRY KREISINGER.

INTRODUCTION.

This bulletin contains a statement of certain results that will be embodied in a report describing investigations of the combustion of fuel made by the United States Geological Survey and the Bureau of Mines in a specially constructed long furnace. The furnace forms part of the fuel-testing plant at Pittsburgh, Pa., which was established and equipped by the Geological Survey, but passed under the control of the Bureau of Mines on July 1, 1910, when a transfer of fuel-testing investigations, authorized by act of Congress, became effective. The work discussed in this bulletin was done under the direction of the Geological Survey.

The furnace was designed and built for an experimental study of the processes of combustion, this study being part of a comprehensive plan for testing fuels to determine their heat value and the manner in which they can be used to best advantage. Although the main object of the researches made with the furnace by the Geological Survey was to examine critically the production of sensible heat by combustion along the path of the gases rising from the fuel, an interesting as well as important side problem developed in the study of the simultaneous dissipation of the heat through the walls and roof of the furnace. Thus incidental data were collected on the temperature gradient through the walls at several places. These temperature data (together with the heat conductivity of the material of the walls) formed the basis for calculating the heat dissipated through the walls.

The object of this report is to present and discuss these temperature data. The discussion particularly concerns the air-space type of wall construction as compared with the solid brick wall or walls type, in which the air space is filled with some solid material of low heat conductivity. The conclusion of the authors, which perhaps will surprise some readers, is that, so far as loss of heat is concerned, a solid wall of brick or any ordinary material is preferable to a hollow wall of the same total thickness, especially if the air space in the hollow wall is near the furnace side.

There is a general belief that since air is a poor conductor of heat, air spaces built into the walls of a furnace will prevent or reduce heat dissipation through the walls. Although there may be instances of

furnace walls in which such construction reduces the rate of heat flow through them, yet as a rule the effect of the air space is just the opposite. While the heat does travel very slowly through the air by conduction, it leaps over the air space readily by radiation. Although this latter mode of heat propagation is common in nature, the laws governing it are not generally known and are seldom taken into consideration when furnace walls are being designed.

It may be stated here that the quantity of heat passing through a portion of a solid wall by conduction depends on the difference between the temperatures of the two planes limiting the portion of the wall. The quantity of heat that passes across the air space in the wall depends on the difference of the fourth powers of the absolute temperatures of the surfaces inclosing the air space. It follows that, in case the heat passes by conduction through the solid portion of the wall, the loss remains approximately the same so long as the temperature of the two limiting planes remains constant, no matter what may be the temperature of the two planes. On the other hand, the heat passing across the air space by radiation increases rapidly with the temperatures of the inclosing surfaces, although the difference between these temperatures may remain constant. This feature will be shown by curves in the latter part of this paper and thoroughly discussed. The important point is that the air space, which is advantageous in the walls of a refrigerator because the temperatures are low, is objectionable in a furnace wall because the temperatures are high.

It is customary to put air spaces in furnace walls between the fire-brick linings and the common brick. Usually the fire-brick lining is only half a brick thick, which construction brings the air space too close to the furnace. The result is that the temperatures of the surfaces inclosing the air spaces are too high, and in consequence too much heat is radiated across the spaces. The heat passing through such walls would be much reduced if the air spaces were filled with brick, or, better, with some cheap nonconducting materials, such as ash, sand, mineral wool, etc. Even where the fire-brick lining is one brick thick (9 inches), the temperature in the furnace may be high enough to raise the temperatures of the air-space surfaces so much that the heat radiated across the space will amount to more than would the heat conducted through a filling, were the filling only common brick. This last statement is amply justified by the data here presented.

ACKNOWLEDGMENTS.

The authors wish especially to thank John K. Clement, physicist, and his assistant, L. H. Adams, both now of the Bureau of Mines, for their aid in devising methods, taking many of the observations, and assisting the authors in studying the data.

EQUIPMENT AND METHODS.

CONSTRUCTION OF FURNACE.

The specially constructed furnace is about 43 feet long over all. At one end is a mechanical stoker; at the other the gases from the combustion chamber discharge into a water-tube boiler. The combustion chamber is a tunnel 3 feet wide, 3 feet 3 inches high, and about 35 feet 6 inches long, with double walls and arch roof. The inner walls and the inner arch are 9 inches thick and are of best quality fire brick. The outer walls, 8 inches thick, are of common and pressed brick. The outer arch, 4 inches thick, is of repressed brick. In the sides a 2-inch air space separates the two walls; in the roof a 1-inch layer of flake asbestos separates the two arches. Air leakage through the walls and roof is minimized by using blowing and exhaust fans to keep the interior of the chamber as nearly as possible at atmospheric pressure.

POSITION OF THERMOCOUPLES.

The data presented in this bulletin consist of three sets of measurements of temperatures at four different depths in a side wall of the furnace, and also of two sets of measurements in the roof. The material of the outer wall and arch is, for convenience, designated common brick in the discussion of the data.

Figure 1 presents the elevation and the plan of the special furnace as it was at the time of getting the data given in this report. Figure 2 gives vertical sections of the furnace. Figure 3 shows the positions of the thermocouples in the side wall at places denoted in figure 1 by *a*, *b*, and *d*. Figure 4 gives the construction of the arch or roof of the furnace and the location of thermocouples at places *c* and *e*. No couples were inserted in the roof corresponding to location *a* in the side wall, because it was feared that the lower side of the lining had been melted away considerably.

The thermocouples were placed in $\frac{3}{8}$ -inch holes, which were drilled in the wall and the roof after the furnace was built. The holes were intended to extend within 1 inch of the outer and inner surfaces of the wall and the roof, as indicated in figures 3 and 4. The depths of the holes for couples 1 and 3 could be measured accurately, and therefore these couples were put at known depths, as represented in the figures. The distances of the ends of the holes for couples 2 and 4 from the inner surfaces could not be accurately measured, so that the distance given in the figures are only approximate. Each couple was placed in such a way that the fused junction touched the bottom of the hole. Near the outer surface of the wall the annular spaces around the wires of the couples were filled with asbestos packing as deeply as possible, in order to prevent radiation of heat from the

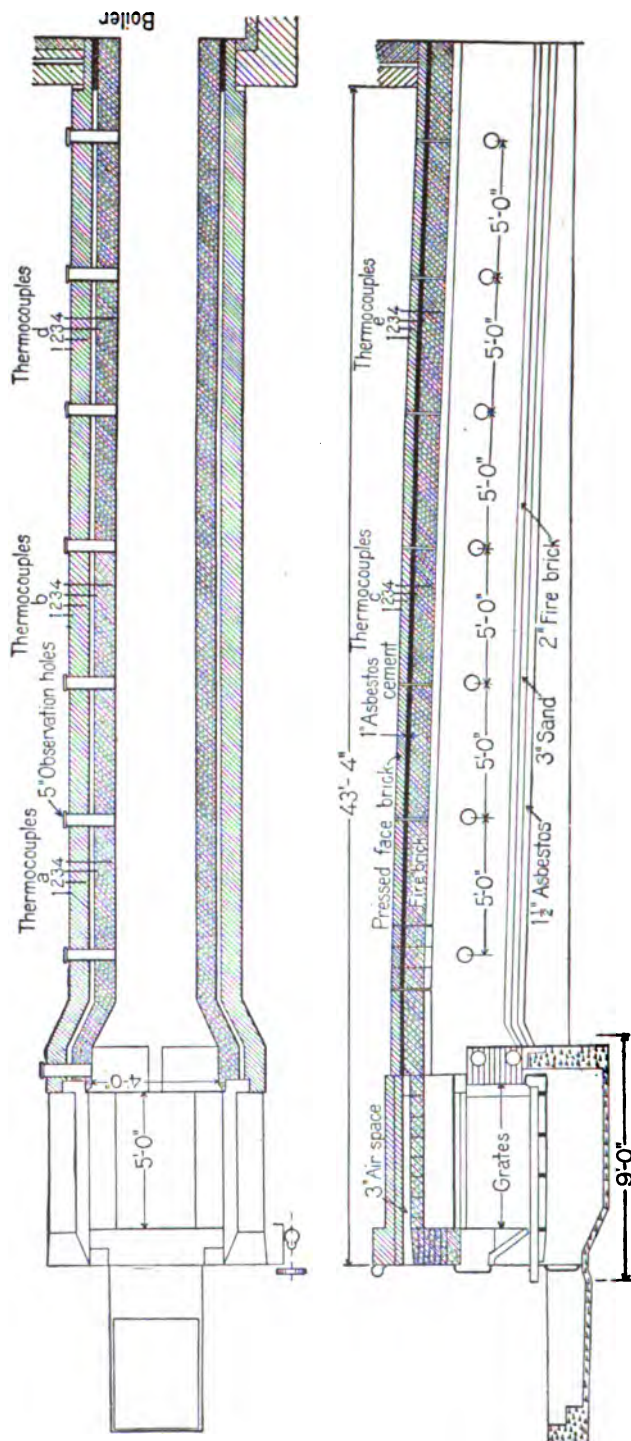


FIGURE 1.—Elevation and plan of furnace.

couples and from the bottom of the holes, and also to stop the cooling of the couples by any current of cold air that might be drawn around the couples into the furnace. This precaution was more effective with thermocouples 1 and 2 than with thermocouples 3 and 4, because the packing could not be put around the couples in the inner fire-brick wall. Consequently, the indications of these couples, and particularly of thermocouple 4, are not exact and should always be considered as approximate.

CONSTRUCTION OF THERMOCOUPLES.

The thermocouples were made of wires about 1.5 mm. ($\frac{1}{16}$ inch) square, and were obtained from the Hoskins Manufacturing Co., of Detroit, Mich. The terminals of the thermocouples were soldered to copper leads, the joints being far enough away from the embedded junctions to insure their being approximately at atmospheric temperature. All other connections in the leads were soldered. Each couple was so connected that it could be thrown into series with a galvanometer by a snap switch. Consequently it was possible to get the twenty readings within about a minute. The couples proved durable and dependable up to $1,400^{\circ}\text{C}$. ($2,552^{\circ}\text{F}$.) and were not much changed by exposure at such temperature to furnace gases and dust.

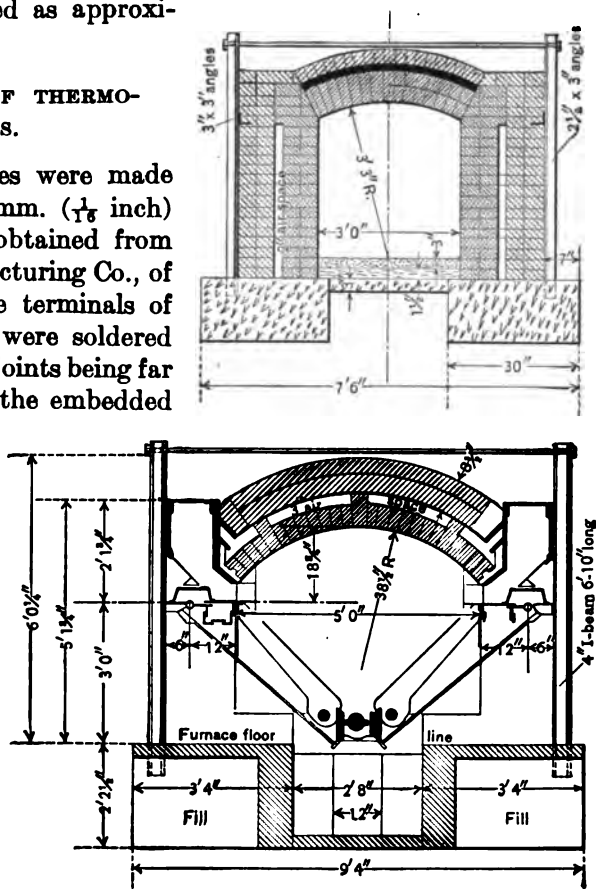


FIGURE 2.—Vertical sections of furnace, through stoker and through long combustion chamber.

READINGS.

Inasmuch as all readings were relative, no attempt was made to get the temperatures closer than 2 or 3 degrees; moreover, any greater accuracy in the couples and instruments would have been

useless, because the bricks containing the holes might have been more porous than others, or some of the holes might have opened into cracks; the holes could not well be inspected. Therefore, the worth of the readings lies as much in their rates of change as in their actual values at any time.

The sets of thermocouples in the side wall are designated *a*, *b*, and *d*, and the sets in the roof are designated *c* and *e*. The couples in each set are numbered from 1 to 4. The reading of any thermocouple is referred to by the letter of the set, with the number of the couple as subscript.

The temperature readings of the thermocouples embedded in the wall and roof were taken on a number of tests.

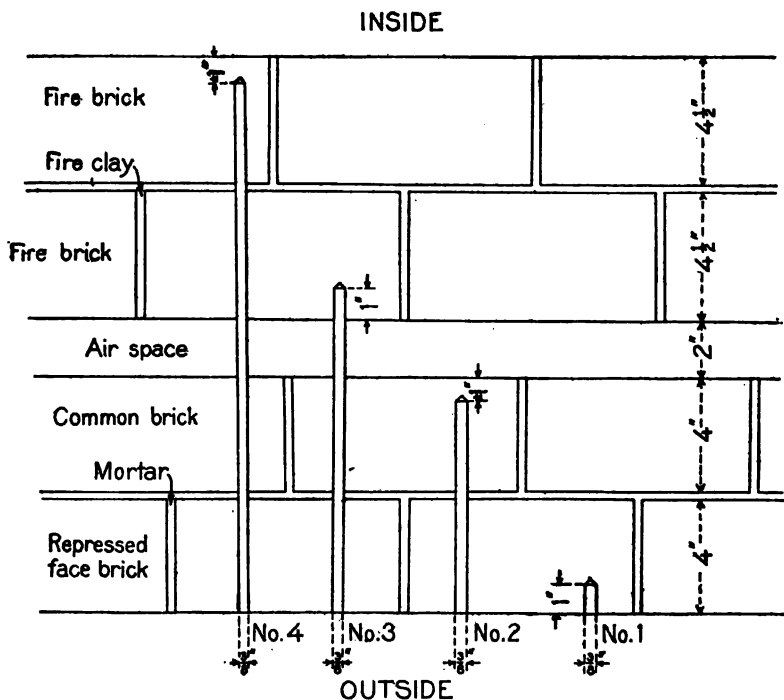


FIGURE 3.—Construction of furnace walls and position of thermocouples.

DISCUSSION OF TEMPERATURE CURVES.

The readings of test No. 16 are platted in figures 5, 7, and 9. This test is a fair representative of other tests in which complete sets of wall and arch readings were obtained every 20 or 30 minutes through the entire test of 29 hours' duration. The test was run until the wall temperatures reached equilibrium, that is, until they ceased to increase. The readings were platted on time as abscissæ, so that the slopes of the curves show the rate of change.

Figure 5 shows the temperature readings for the entire test of the thermocouple sets placed at *b* and *c*. (See fig. 1.) This chart shows

that, excepting perhaps within 1 or 2 inches from the inside of the furnace, the brickwork is nearly cold when the test is started, and that as the heat flows out through the wall from the inner surface the temperatures of the portions of the wall farther from the furnace rise for about 19 hours of the test, after which period the equilibrium of wall temperature is reached. Up to this time most of the heat entering the wall goes to raise the temperature of the latter; thereafter all the heat entering the wall goes through it and is lost by radiation.

EQUILIBRIUM OF FURNACE-WALL TEMPERATURES.

In the following discussion the nearly true assumption is made that there is no cooling effect due to leakage currents of air through the brickwork or into, out of or along the air space.

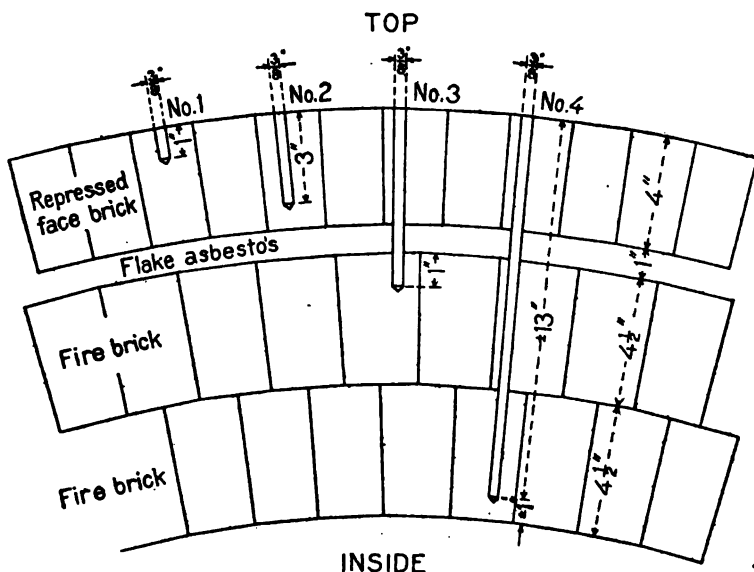


FIGURE 4.—Construction of furnace roof and position of thermocouples.

During the equilibrium, the quantity of heat passing through an inner part of the wall is exactly equal to the heat going through another part farther out. For example, the quantity of heat which is conducted through the inner fire-brick wall is exactly equal to the heat which passes across the air space, and is exactly equal to the heat which is conducted through the outer common brick wall, and also equal to the heat radiated or taken in any other way from the outside surface. If this equality of heats did not exist, the equilibrium would be impossible. For example, let it be assumed that more heat passed through the inner than through the outer wall and over the air space; then the heat would accumulate near the surface next to the air space and the temperatures shown by thermocouples 3 would be rising, a circumstance opposed to equilibrium. Again, let it be

assumed that more heat passed through the outer wall than through the inner one and through the air space; in that case the heat in the outer wall near the surface next to the air space would diminish and the temperatures shown by thermocouples 2 would drop, an event that would be contrary to the equilibrium conditions. The statement that during equilibrium of the wall temperatures in any cross section of the wall the quantity of heat passing through one part of the section is exactly equal to the quantity passing through any other part is therefore justified.

TEMPERATURE DROPS IN FURNACE WALLS.

The quantity of heat flowing by conduction from one plane to another through any portion of the furnace wall depends upon the difference of temperature between these two planes and upon the resistance to the heat flow. With the same temperature difference, if the resistance is high, a small quantity of heat flows through; if the resistance is low, a large quantity flows through. Or, if the quantity of heat is to remain constant the temperature difference must be large if the resistance to the heat flow is high, and small if the resistance is low.

So, in the case of the furnace wall, where the quantity of heat passing between any two planes which are parallel to the surfaces of the walls is the same, the temperature difference between any two planes indicates the resistance which the material or space between the two planes offers to the flow of heat. For example, if the temperature difference between the faces of the fire-brick wall is high, it may be said that the resistance to the heat flow of the fire-brick wall is high; or, if the temperature difference between the two surfaces on each side of the air space is low, it may be inferred that the resistance to the heat passage across the air space is low. Thus it is possible to rely on the temperature difference as being a true indicator of high or low resistance to heat flow between any two planes which are parallel to the surface of the wall. With this knowledge the reader can turn to the charts and study the resistances of the fire brick, the air space, the asbestos layer, and the common brick, and the relative value of these materials as heat insulators in the construction of furnace walls.

Figure 6 gives the temperature drops through the side wall as recorded by the set of couples placed at *b*, and through the roof as recorded by the set of couples placed at *c*. At the foot of the figure is shown diagrammatically the thickness of the side wall, and at the top of the figure is shown the thickness of the roof; in each case the measurements of thickness are used as abscissæ in the chart. The temperatures at the various points are plotted as ordinates. The figure shows three temperature gradients or drops through the wall and through the roof, one at 11 a. m., April 12, when the test was started, one at 4 p. m. the same day, and one at 2 p. m. the next day. The first two gradients give the relation of the temperatures before

the equilibrium is reached, and are interesting only when compared with one another to show how the temperatures change with respect to each other while the walls are being heated. The last gradient represents the equilibrium and is of most interest.

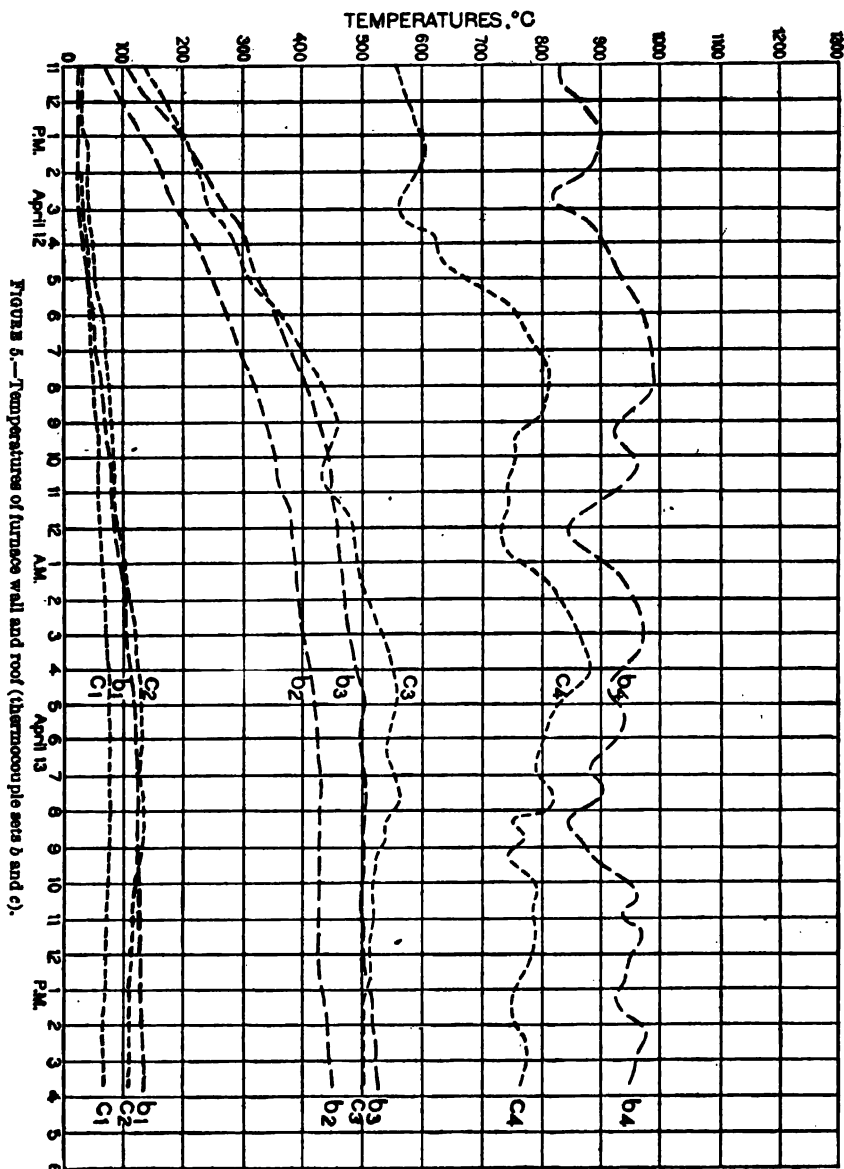
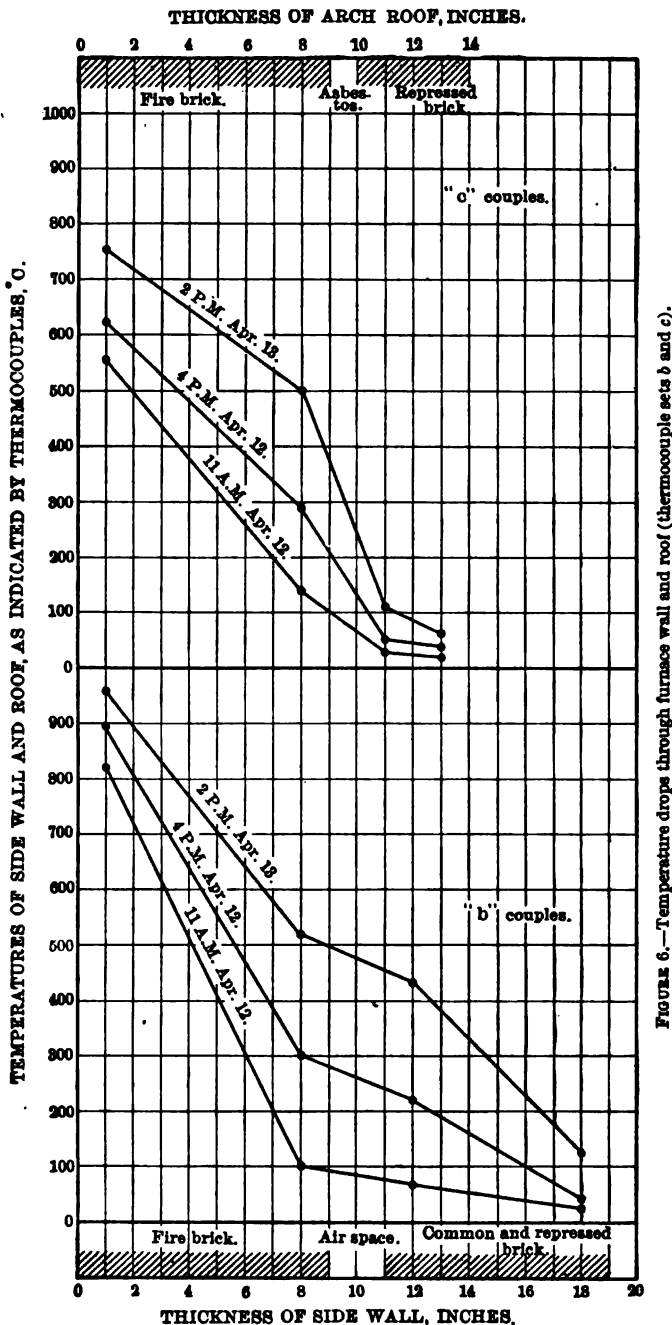


FIGURE 5.—Temperatures of furnace wall and roof (thermocouple sets *b* and *c*).

RESISTANCE TO HEAT FLOW OF FIRE BRICK, AIR SPACE, ASBESTOS, AND COMMON BRICK.

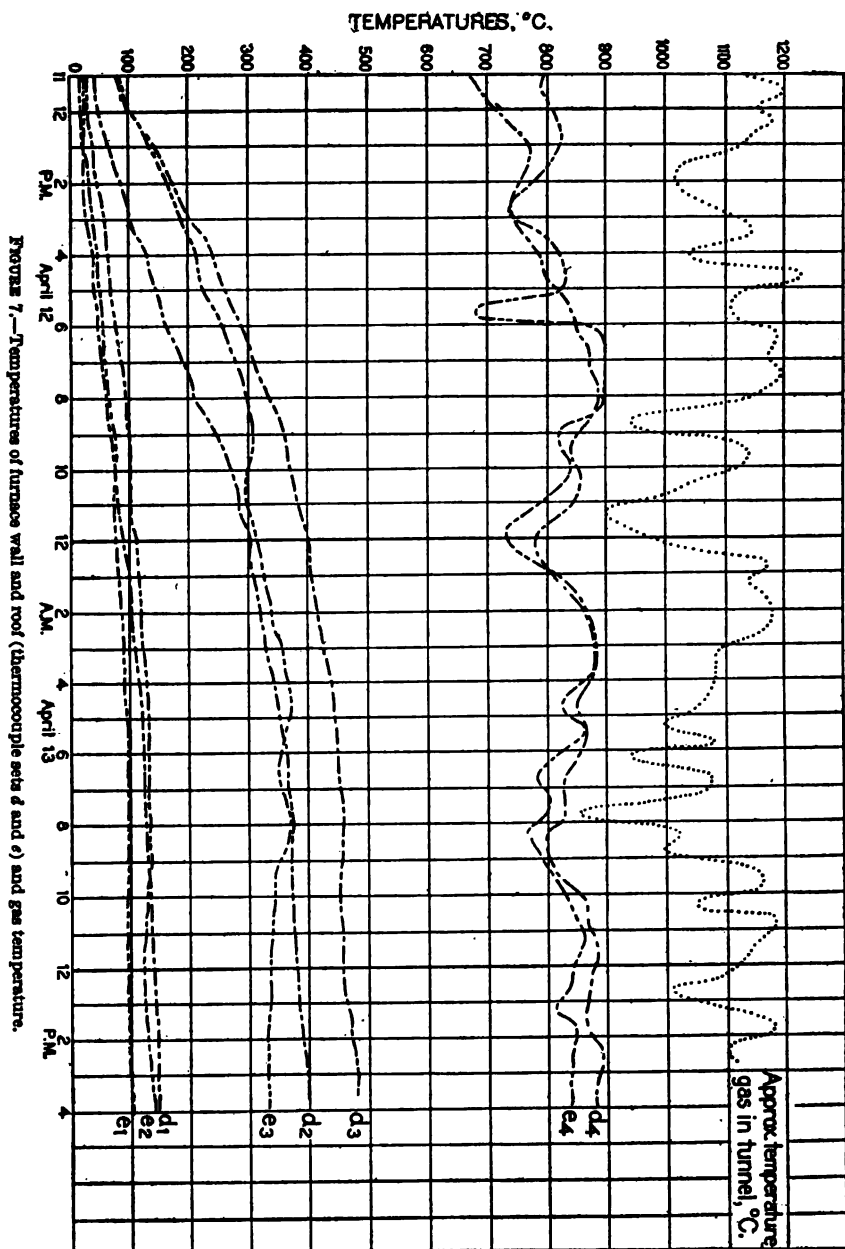
The striking feature concerning the side-wall thermocouples, set *b*, is the large temperature drop through the fire-brick wall, the very small drop through the air space, and, again, the large temperature

drop through the common brick wall. These drops plainly indicate that the resistance to heat passage of the air space is very low compared with that of either brick wall, only about one-fourth as much. The last temperature gradient through the roof, as given by



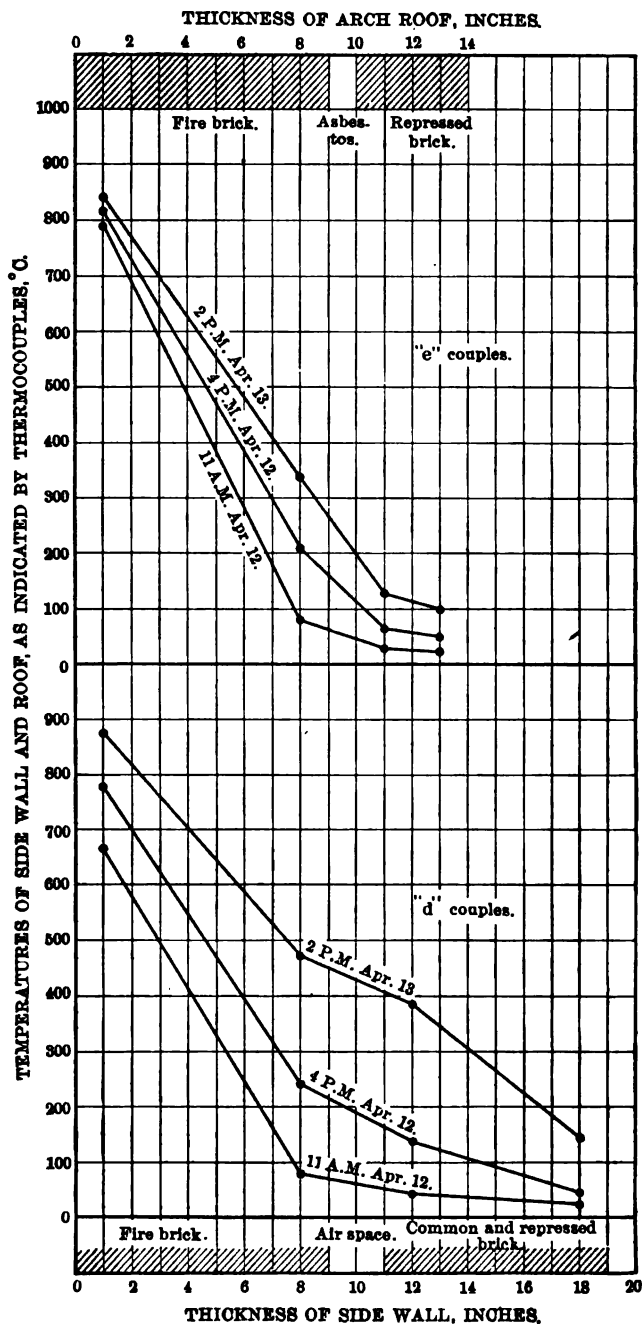
pared with that of either brick wall, only about one-fourth as much. The last temperature gradient through the roof, as given by

the set of thermocouples *c*, shows a rather low temperature drop through the fire brick, a high drop through the 1-inch layer of asbestos, and a rather small drop through the common brick. These tempera-



ture drops indicate that the resistance to heat flow of the 1-inch asbestos layer is higher than that of 7 inches of fire brick. By comparing the last gradient of couples *b* with that of couples *c*, it is easy

to see that 1 inch of asbestos is much more effective as a heat insulator under the existing conditions than a 2-inch air space. Although the



total thickness of the roof is 5 inches less than that of the side wall, a smaller quantity of heat per square foot is lost through it than

through the side wall. The relative amounts of heat passing through the two constructions can be figured approximately in the following manner:

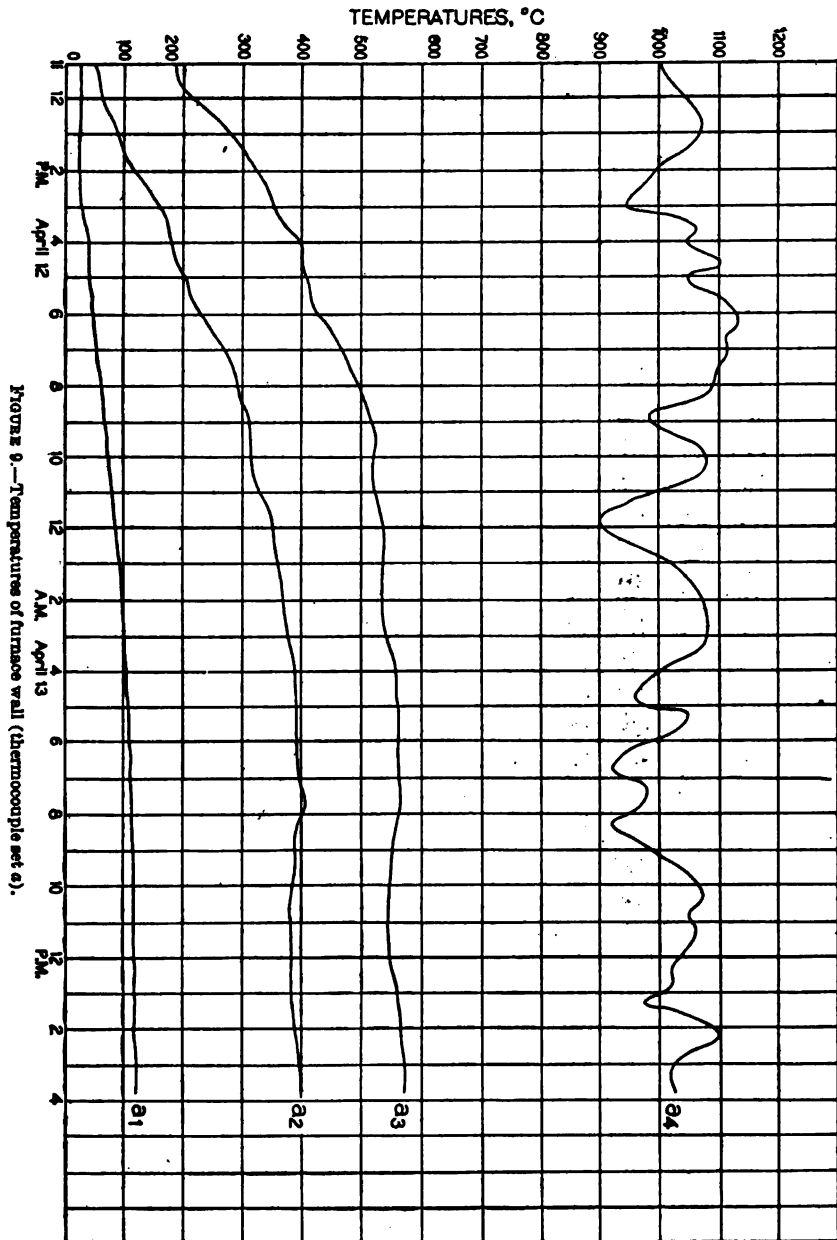


FIGURE 9.—Temperatures of furnace wall (thermocouple set a).

Inasmuch as the fire brick in the roof are made of the same material as the fire brick of the side wall, the conductivity of the fire brick in both places may be taken to be the same; and as the thicknesses

are the same, the amounts of heat flowing through both per square foot per unit of time are proportional to the temperature drops. Thus the temperature drop through the fire brick in the side wall is from 965°C. to 510°C. , or 455°C. ; the temperature drop through the similar brick in the roof is from 750°C. to 500°C. , or 250°C. The heat lost through the side wall is $\frac{455}{250} = 1.8$ times as much as the heat lost through the roof.

In a similar manner one can compute the relative heat losses from the temperature drops through the common brick. In the side wall this drop through 6 inches of common brick is from 440°C. to

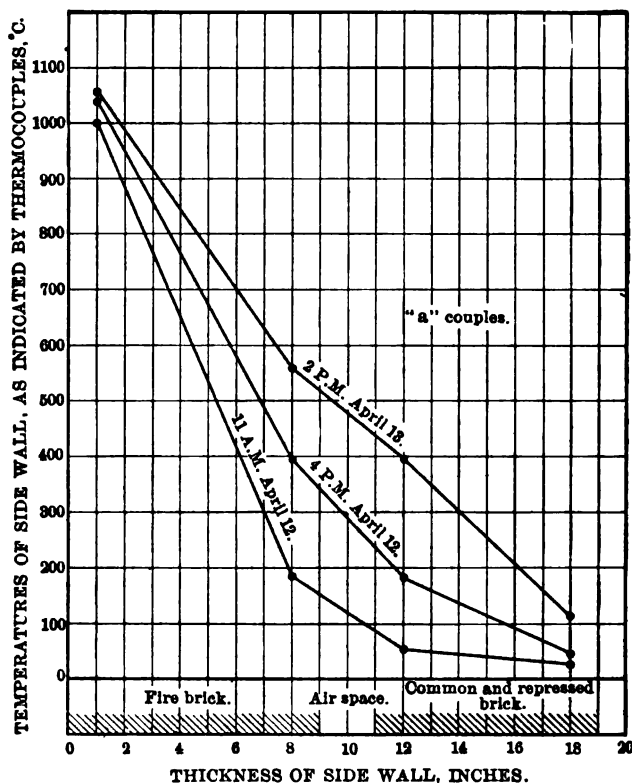


FIGURE 10.—Temperature drops through furnace wall (thermocouple set a).

130°C. , or 310°C. ; and in the roof this drop through 2 inches of common brick is from 115°C. to 65°C. , or 50°C. , which, when corrected to the equivalent drop for 6 inches of common brick, gives 150°C. According to these figures, the amount of heat lost through the side wall is $\frac{455}{150} = 3$ times as great as through the roof.

Figure 7 gives the temperature readings for the entire test (No. 16) of the thermocouple sets located at *d* and *e* (see fig. 1), and also the temperature readings inside the furnace as measured by a Hoskins thermocouple placed in the center of the furnace. The chart shows

that in general the temperature inside the furnace and the temperature indicated by thermocouple *a*, rise and fall together, as may be expected, the temperature of the wall lagging somewhat.

Figure 8 gives the temperature gradients at the places *d* and *e* at exactly the same time as those given in figure 6. The indication of these gradients confirms that of the gradients of figure 6, which is that under the given conditions the air space offers decidedly less resistance to the heat flow than either brick wall and much less than the 1-inch layer of asbestos. The relative heat losses through the two constructions as indicated by the temperature drops of figure 8 are as follows:

Through the fire-brick portion of the side wall the temperature drop is from 880° C. to 475° C., or 405° C.; through the similar portion of the roof the temperature drop is from 840° C. to 340° C., or 500° C. According to these temperature drops, the amount of heat passing through the side wall is $\frac{405}{500} = 0.8$ of the heat passing through the roof. This small ratio is undoubtedly due to the incorrect temperature indication of thermocouple 3 in set *e*. Upon examining the upper half of figure 8, it is apparent that couple 3 reads too low. Very likely, when the thermocouple was being placed some of the flake asbestos fell into the hole, partly filling it, so that the junction of the couple did not come into contact with the fire brick but with the asbestos, which insulated it from the brick. There might also have been a crack in the brick, where the hole was drilled, through which air was drawn into the furnace around the thermocouple, thus cooling it. Or, the hole might have been drilled into mortar between two bricks, which mischance would also tend to lower the temperature of the thermocouple in question. There was no way to inspect the bottom of the hole after it was drilled and the couple placed in it. Whatever the cause, it is quite certain that the temperature indicated by thermocouple 3 is about 200° C. too low.

Even with this too low temperature of couple 3, the drop through the asbestos layer is much greater than through the air space and greater than through the common brick. The relative quantities of heat flowing through the roof and the side wall as figured from the temperature drops through the common brick are about as follows:

In the side wall the temperature drop through 6 inches of common brick is from 390° C. to 140° C., or 250° C.; in the roof the temperature drop through 2 inches of common brick is from 130° C. to 100° C., or 30° C., which, when corrected to the equivalent for a 6-inch wall, is 90° C.; the heat flowing through the side wall is $\frac{250}{90} = 2.8$ times as much as the heat flowing through the roof.

It should be borne in mind that the temperature indications are only approximate and therefore the relative heat losses are necessarily only approximate. However, three of these approximate

computations indicate that the heat flowing through the side wall is about twice as much as that flowing through the roof, and only one

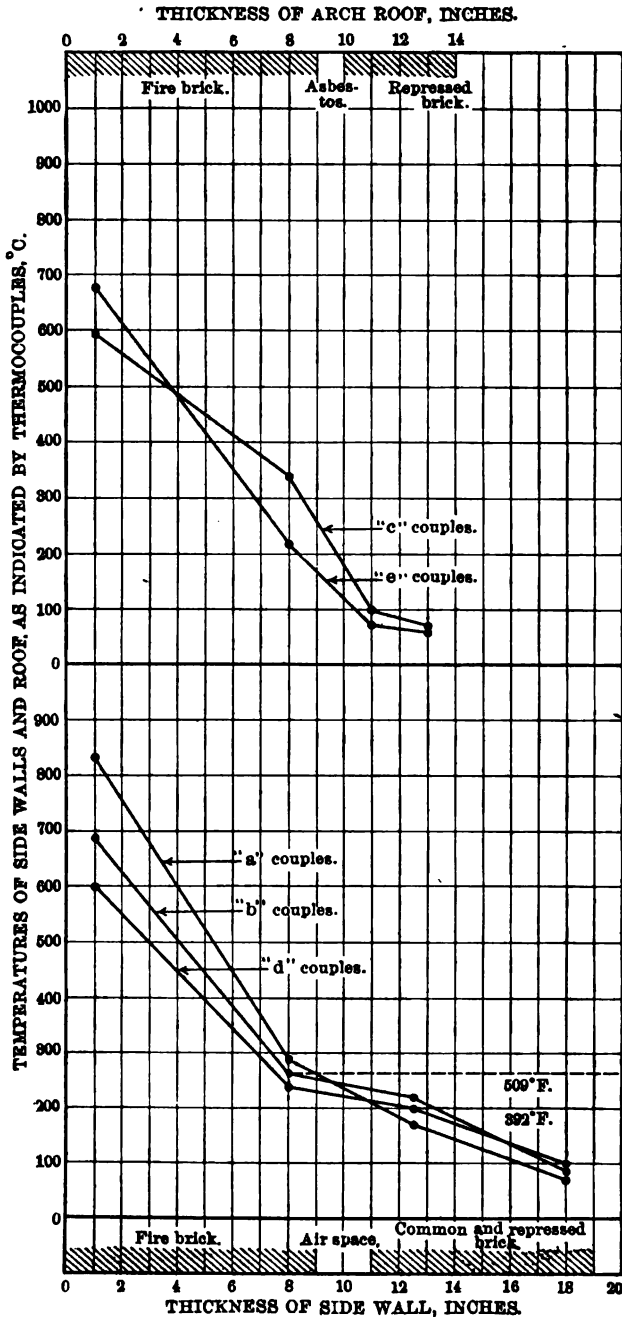


FIGURE 11.—Temperature drops through furnace wall and roof (thermocouple sets a, b, c, d, and e).

indicates that the quantities of heat are about equal; besides, it is fairly certain that the data for this one computation are incorrect.

Figure 9 gives the temperature readings of the four thermocouples located at *a*. Figure 10 shows the temperature gradients as indicated by the four thermocouples *a*. The time of the temperature readings used to draw these gradients is the same as for the gradients in figures 6 and 8. The same striking feature is shown in figure 10 as in the latter two figures; that is, the temperature drop across the air space is much smaller than the drop through either brick wall, indicating that the resistance of the air space to the heat flow is lower than that of solid brick. The figure does not contain corresponding temperature gradients for the roof, as the temperatures were not recorded.

In figure 11 are given the temperature gradients of all five sets of thermocouples for test No. 17. All the temperature readings were taken within a minute, so that the readings may be considered as simultaneous. Each gradient is labeled with a letter referring to the particular set of thermocouples with which temperatures were read. The indication of these gradients confirms that of the gradients obtained from test No. 16, which is that the resistance of an air space to heat flow is lower than that of either common or fire brick and much less than that of asbestos. The average temperature drop through the fire-brick portion of the roof is about 350° C., and that through the side wall about 450° C. According to these temperature drops, the amount of heat flowing through the side wall is $\frac{450}{350} = 1.3$ times that passing through the roof.

Attention is again called to the fact that the reading of thermocouple *e*, is incorrect, and, for reasons previously given, very likely too low. Nevertheless, the readings as given are greatly in favor of the solid insulating material and against the air space.

CONCLUSIONS.

The results of the investigation as outlined in this bulletin justify the following conclusions:

In furnace construction a solid wall is a better heat insulator than a wall of the same total thickness containing an air space. This statement is particularly true if the air space is close to the furnace side of the wall, and if the furnace is operated at high temperatures. If it is desirable in furnace construction to build the walls in two parts, so as to prevent cracks being formed by the expansion of the brickwork on the furnace side of the walls, it is preferable to fill the space between the two walls with some "solid" (not firm, but loose) insulating material. Any such easily obtainable materials as ash, crushed brick, or sand offer higher resistance to heat flow through the walls than an air space. Furthermore, any such loose material by its plasticity reduces air leakage, which is an important feature deserving consideration.

DISCUSSION OF PHYSICAL LAWS.

THE LAW OF RADIATION.

The object of the following paragraphs is to explain, by the application of well-demonstrated physical laws, why the results presented in the foregoing pages came out as they did.

It has been stated that although air is a poor conductor of heat, air spaces in furnace walls are not desirable, because the heat leaps across the space by radiation. This latter mode of heat travel is very common in nature; for instance, the source of practically all the energy on the earth, developed and undeveloped—the heat which the earth gets from the sun—comes to it by radiation. Notwithstanding the commonness of this phenomenon, the true laws of radiation remained unknown to science until two or three decades ago, and even at present they are becoming recognized only slowly by the engineering profession.

In the past, when radiation was considered in any engineering problem, the first-power law was usually used in calculation. This faulty law, which was proposed by Isaac Newton, stated that the heat radiated from a hot body to a cold surrounding body was proportional to the difference of their temperatures. About thirty years ago Stefan found, however, that at high temperatures Newton's law was wrong, and that the radiation was very nearly proportional to the difference of the fourth powers of the absolute temperatures of the two bodies. Some years later Boltzmann demonstrated mathematically that from the principles of thermodynamics the fourth-power law should hold exactly for an ideal black body. There are, however, no substances in nature which behave exactly like the black body. The sooted surface comes very close to it (within 2 to 5 per cent, depending on the kind of soot) and can be taken for all practical purposes as a standard. The fourth-power law, which is known as the Stefan-Boltzmann radiation law, is expressed by the following equation:

$$(1) \quad H = C(T_1^4 - T_2^4)$$

where H = the net heat exchanged between the hot and cold surface per unit of the hot surface per unit of time.

T_1 = the absolute temperature of the hotter surface.

T_2 = the absolute temperature of the colder surface, which surrounds the hot surface.

C = a constant depending on the units of area and time, on the unit in which the heat is measured, and on the scale in which the temperatures are expressed.

If H is expressed in small calories per square centimeter of the hot surface per minute and T_1 and T_2 are expressed in degrees centigrade on the absolute scale, then

$$(I) \quad C = 7.65 \times 10^{-11} = \frac{7.65}{100,000,000,000}$$

If H is expressed in B. t. u. per square foot of the hot surface per minute and T_1 and T_2 are expressed in degrees Fahrenheit on the absolute scale, then

$$(II) \quad C = 2.66 \times 10^{-11} = \frac{2.66}{100,000,000,000}$$

The above constants are good only for sooted surfaces when the hotter surface is entirely surrounded by the cooler surface, the condition being that the hot surface must not "see" anything but the cold surface. This condition can be satisfied by either of the two cases shown in figures 12 and 13.

In figure 12 the hot surface is the outer surface of a spheroidal body that is inclosed within a larger spheroid, the inside of which forms the cold surface. Any square centimeter, A , of the hot surface can not "see" anything but the cold surface.

On this spheroidal body two areas, A , are indicated, with the intention of illustrating the fact that the radiating surface may be either plane or convex without changing the amount of heat radiated, but can not be concave, for then some portions of surface A would "see" other hot surface of its own hot body as well as the cold surface.

In figure 13 the hot surface is a plane; the cold surface is also a plane parallel to the first one and infinitely large, so that any square centimeter, A , of the hot surface, A , can not "see" anything but the cold surface; that is, the angle of "vision" approaches the spherical angle of 180° .

In figure 14 the above conditions are not satisfied, because any square centimeter, A , of the hot surface can "see" something else besides the cold surface; that is, the angle of "vision" is less than a spherical angle of 180° , so that the cold surface does not receive all the heat radiated by any portion of the hot surface.

Heat passing from portion A through angles α and c misses the cold surface entirely. Therefore, in such instances as are pictured by figure 14 the values for C must be calculated for each solid angle of "vision" to each respective cold surface's temperature and coefficient of radiation.

Several curves (figs. 15-18) have been platted to show graphically the significance of the Stefan-Boltzmann law.

The law of radiation and the conditions as presented in the preceding paragraphs apply to sooted surfaces only. For surfaces not sooted this law must be somewhat modified. For instance, a brick surface does not radiate so much heat as a sooted surface of the same temperature, nor does it absorb so much of the heat which reaches it by radiation. In consequence, the net heat exchanged between two brick surfaces is less than that exchanged between two sooted surfaces at the same temperatures. The ratio of the two quantities of the net heat exchanged is here called the "coefficient

of net heat exchange." The latter increases with the absolute temperature of the surfaces. For the brick surfaces this coefficient at 700°C. , absolute, is perhaps 0.5; that is, two brick surfaces will exchange only about 0.5 of the net amount of heat exchanged between two sooted surfaces which are at the same respective temperatures as the brick surfaces.

The conditions existing between the two surfaces of an air space in a furnace wall are very nearly like those shown in figure 13. The

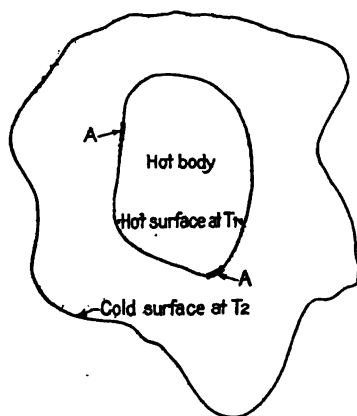


FIGURE 12.—Radiation of heat from hot spheroidal surface to inclosing cooler surface.

two surfaces of the air space are exposed to nothing excepting to one another.

By multiplying the values of the constant C as given on page 20 by this coefficient for brick, formula (I) can be applied directly to compute the heat radiated across an air space, or the same results can be obtained if the values of H figured for sooted surfaces are multiplied by 0.5.

Figuring the net heat radiated between any two surfaces by formula (I) is rather cumbersome; therefore figures 15 to 18, inclusive, have been worked out to give the net heat radiated from one sooted surface to another, and satisfy in all instances the conditions that are shown by figures 12 and 13. To obtain the net radiation between any other substances the values of H obtained from the charts are multiplied by the proper coefficients of net heat exchange, which in all cases are less than 1.00. Figures 15 and 16 are worked out in metric and figures 17 and 18 in English units. Figure 15 is a part of figure 16 on a larger scale, the range of the former being limited to the portion of the

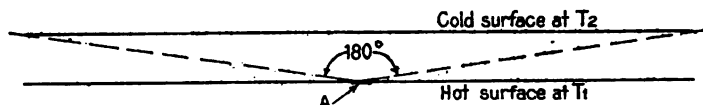


FIGURE 13.—Radiation of heat from infinite hot plane surface to infinite cooler plane surface.

latter below the heavy broken line. In like manner figure 17 is a part of figure 18 on a larger scale, the part below the 500°F. line.

To illustrate the use of the charts, let it be required to find the radiation from a sooted surface at 700°C. absolute (427°C. on the ordinary scale) to a sooted surface at 550°C. absolute (277°C. ordinary scale). Turn to figure 15. At the foot of the chart is the absolute temperature of the cooler surface in degrees centigrade.

Take the vertical line starting from a temperature point of 550°C . and follow it to the curve labeled 700°C . This is the temperature of the hotter surface. From the intersection of the vertical line with the curve draw a horizontal line to the scale on the left hand, which scale indicates that in the assumed case the radiation is 11 small calories per minute per square centimeter of the hot surface. If the radiation for brick surfaces not sooted is wanted, multiply 11 by 0.5, say; the result is 5.5 calories, which is the approximate net radiation from 1 square centimeter of the hotter brick surface when its absolute temperature is 700°C . and that of the cooler surface is 550°C .

Again, suppose that it is desired to know the radiation from a sooted surface at $1,600^{\circ}\text{C}$. to a sooted surface at $1,200^{\circ}\text{C}$. At the foot of figure 16 take the vertical line starting from $1,200^{\circ}\text{C}$., follow it until the curve of $1,600^{\circ}\text{C}$. is reached, and then follow the horizontal line passing through the intersection point to the scale on the left; the scale indicates that in the given case the net radiation is 340 calories per square centimeter of the hotter surface. If the radiation from a brick surface under the foregoing conditions is wanted, multiply 340 by 0.5, and the result is 170 calories for brick not sooted.

To take a more specific example, use figures 17 and 18 in figuring the rate of heat radiation across the air space in the side wall of the long-combustion chamber for test No. 16. For the temperatures of the surfaces, take the average temperatures of the highest gradients of figures 6, 8, and 10; that is, those gradients which represent equilibrium of temperature.^a From the three figures it is found that the average temperature indicated by the three thermocouples embedded near the hotter surface of the air space is 514°C ., ordinary scale, or $1,418^{\circ}\text{F}$. on the absolute scale, and the average of the indications of the three couples embedded near the colder surface is about 408°C ., ordinary scale, or $1,227^{\circ}\text{F}$., absolute scale. We have then:

$$T_1 = 1,418^{\circ}\text{F}.$$

and

$$T_2 = 1,227^{\circ}\text{F}.$$

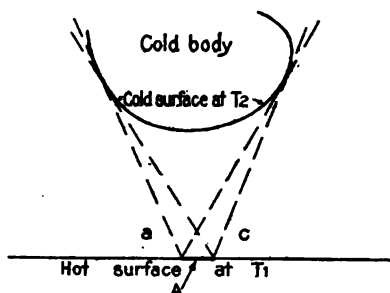


FIGURE 14.—Radiation of heat from infinite hot plane surface to convex surface.

^a It should be remembered, however, that in the experiments discussed the junctions of the thermocouples were embedded in the brick about 1 inch from the surface, and therefore the assumed temperatures are farther apart than the actual temperatures were.

Necessarily, higher values of the net heat exchanged or radiated will result when these assumed temperatures are used in computation. The exact location of the junctions of the thermocouples could not be determined, and therefore no accurate and reliable correction of the surface temperatures can be made.

The net radiation corresponding to these temperatures can be found approximately from figure 17 in the following manner:

On the scale at the foot of the chart locate the temperature of 1,227° F., absolute, the temperature of the colder surface. This temperature will be found between the points of 1,200° and 1,300° F. After locating this point, follow an imaginary vertical line rising from it and cutting the curve of 1,418° F. This curve, though not on

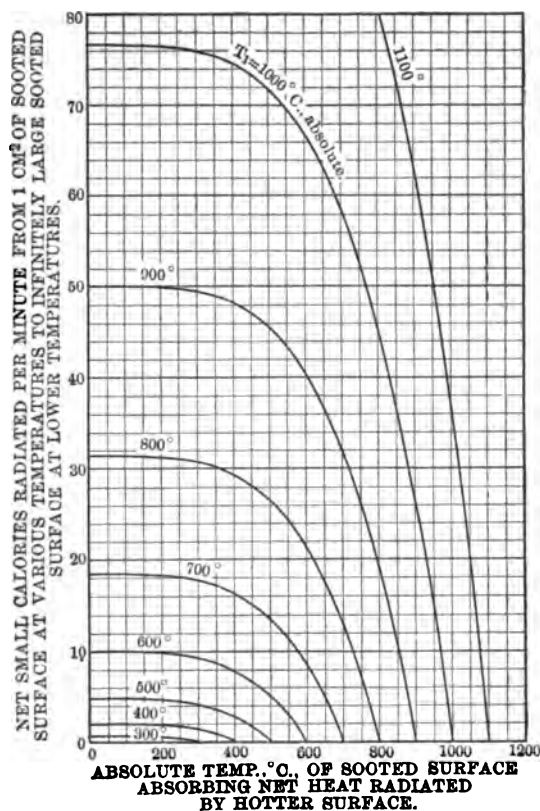


FIGURE 15.—Radiation from a hot surface to an infinite cooler surface at various temperatures; relations expressed in the metric system of units.

the chart, can be imagined in its proper place between the curves of 1,400° and 1,500° F. From the intersection of the two imagined lines, draw a horizontal line to the scale on the left, which gives the heat in British thermal units radiated per minute per square foot of the hotter surface. In the case under consideration, the radiated heat is found to be 44 B. t. u. for sooted surfaces. The radiation for brick surfaces is obtained by multiplying 44 by the coefficient of heat exchange for brick, which is taken as 0.5. The radiation for the brick is then $44 \times 0.5 = 22$ B. t. u.

Now, the wall is 40 feet long and 4 feet high, which makes the total radiating surface

$40 \times 4 = 160$ square feet, and the total radiation through the space in one side wall is $160 \times 22 = 3,520$ B. t. u.

THE LAW OF HEAT CONDUCTION.

The law governing the travel of heat by conduction is simpler than the law of heat travel by radiation and is well known and much used by engineers.

The amount of heat conducted through a unit of area from one part of a body to another is proportional to the temperature dif-

ference of the two parts, proportional to the conductivity of the body, and also inversely proportional to the distance between the two parts of the body. This law is expressed by the following equation:

$$(2) \quad H = \frac{c}{d}(T_1 - T_2)$$

Where H = the quantity of heat conducted per unit of area per unit of time.

c = the conductivity of the material, which varies somewhat with the temperature.

d = the distance between the two parts of the body.

T_1 = the temperature of the hotter part.

T_2 = the temperature of the colder part.

The experiments made by J. K. Clement and W. L. Egy^a at the University of Illinois engineering experiment station show the conductivity of fire brick at 700° C. to be about 0.0024 in the metric units, which is equivalent to 0.1158 B. t. u. per square foot per minute per 1° F. difference of temperature when the distance between the two heat-exchanging planes is 1 inch.

This conductivity being known formula (2) can be used to figure out the quantity of heat passing by conduction through the fire brick portion of the side wall for the test No. 16, and this quantity of heat can be compared with that radiated across the space, as computed on page 24. For T_1 take the average of the three temperatures 1 inch (about) from the furnace as given by the highest three gradients of figures 6, 8, and 10, and for T_2 take the average of the corresponding three temperatures 1 inch from the air space of the fire-brick side.

The values are:

$$T_1 = 962^\circ \text{C.} = 1,763^\circ \text{F.},$$

$$T_2 = 516^\circ \text{C.} = 960^\circ \text{F.},$$

and $d = 7$ inches.

Substituting the above values in equation (2) gives

$$H = \frac{0.1158 \times (1763 - 960)}{7} = 13.3 \text{ B. t. u. per square}$$

foot of the wall.

For one whole side it will be

$$13.3 \times 40 \times 4 = 2,128 \text{ B. t. u.}$$

The heat radiated across the air space was calculated to be 3,520 B. t. u.

Hence, the temperatures used in both calculations being regarded as in equilibrium, the two heats should be equal if the temperatures were correct. However, it is known, and has already been stated, that

the temperatures of the radiating surfaces of the air space as used in the calculation were only approximate, and that they were farther apart than the temperatures of the surfaces actually were. This is the reason why the result of the computation of heat radiated across the space is too high. If allowance is made for this inaccuracy of data and for the fact that the coefficient of net heat exchange could only be approximated from the meager experimental data that is at present available on this subject, the agreement between the amount of

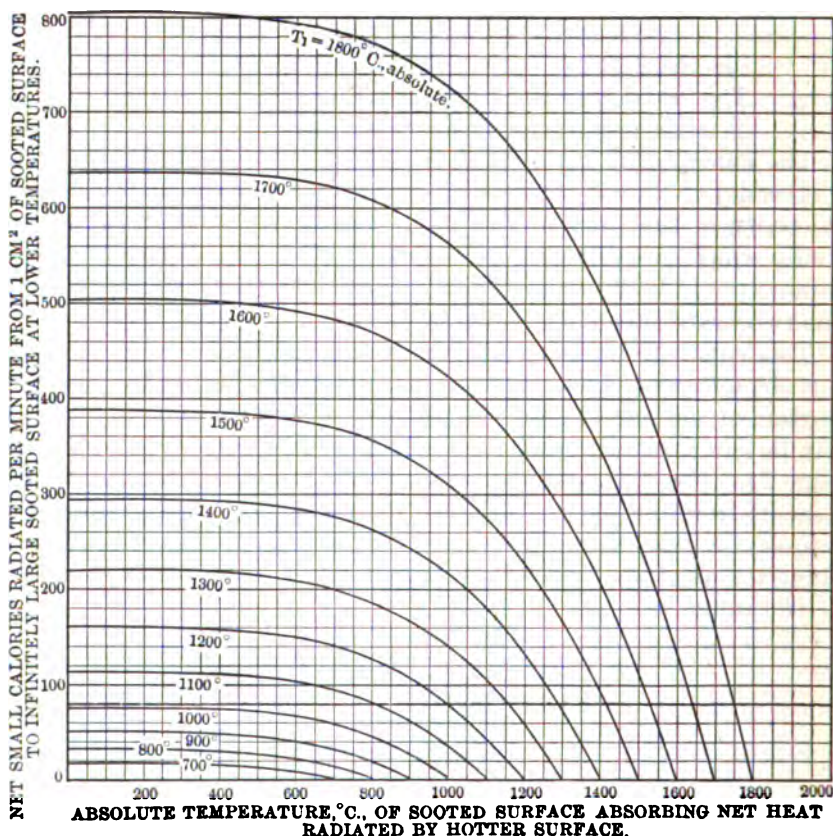


FIGURE 16.—Radiation of heat from a hot surface to an infinite cooler surface at various temperatures; relations expressed in the metric system of units.

heat radiated across the air space and that conducted through the fire-brick wall is better than might be expected. The agreement further shows that the losses of heat by radiation could be figured out in an entirely rational way if more were known about the radiating properties of fire brick and similar material.

It may be stated here that in the specific case of the two given methods of calculating the amount of heat passing through the side wall, the method of calculation by the law of conduction is the more accurate and the more reliable one, because the data for it are more

accurate, and any errors in temperature are not raised to the fourth power, as they are in calculating by the law of radiation.

It will be interesting to figure out the total heat lost through the walls, roof, and bottom of the furnace by the methods of conduction. From previous calculation the loss through both side walls for test No. 16 was

$$2,128 \times 2 = 4,256 \text{ B. t. u.}$$

The average temperature drop through the fire-brick portion of the roof when the temperatures were in equilibrium was from 787° to 420° C., or from $1,449^{\circ}$ to 788° F.

Substituting these values in equation (2) gives:

$$H = \frac{0.1158 \times (1,449 - 788)}{7} = 10.95 \text{ B. t. u. per square foot of the roof.}$$

Now, the roof was 40 feet long and the fire-brick portion about 4 feet wide, which makes the total area

$$40 \times 4 = 160 \text{ square feet}$$

and the total heat lost through the roof

$$160 \times 10.95 = 1,752 \text{ B. t. u. per minute.}$$

On the assumption that the heat lost through the bottom of the furnace is about equal to that lost through the roof, the total heat lost from the furnace by dissipation through the walls, roof, and bottom is

$$2 \times 2,128 + 1,752 + 1,752 = 7,760 \text{ B. t. u. per minute.}$$

The rate of combustion was about 900 pounds of coal per hour or 15 pounds per minute. If each pound of coal developed 12,000 B. t. u., the total quantity of heat passing through the furnace per minute was

$$12,000 \times 15 = 180,000 \text{ B. t. u.}$$

Of this heat $\frac{7,760}{180,000} = 4.3$ per cent is lost in dissipation through the walls, roof, and bottom of the furnace. Probably 1.5 to 2 per cent of the total loss or nearly half the proportional loss could be avoided by proper selection of material and proper furnace-wall construction.

COMPARISON OF THE LAWS.

So far the law of radiation and the law of conduction have been discussed and illustrated separately. It remains to compare or contrast these laws so as to bring out more prominently the influence of the fourth power in the radiation law. Figure 19 has been devised to show such contrast. The upper curve shows how the amount of heat radiated from one sooted surface to another at a constant temperature difference of 100° C. or 180° F. increases as their temperatures rise. The scale at the foot of the chart gives the absolute

temperature of the hotter surface in degrees centigrade, the scale at the top gives the same in degrees Fahrenheit, the scale at the left gives the net heat exchanged between the two surfaces in metric units, while the scale on the right gives the same in English units.

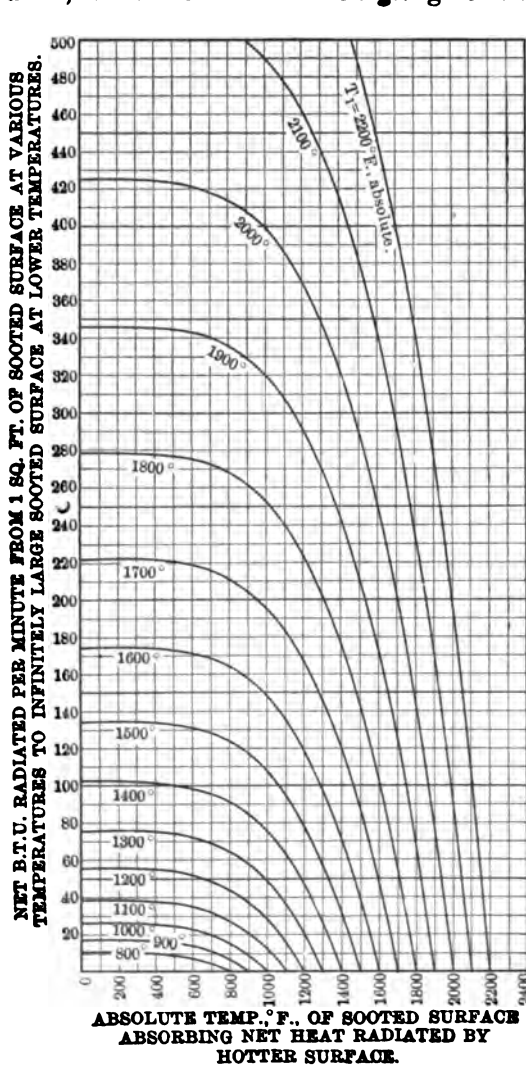


FIGURE 17.—Radiation of heat from a hot surface to a cooler surface; relations expressed in British thermal units, degrees Fahrenheit, and square feet.

The upper horizontal line at the foot of the figure shows the relation of the heat transmitted by conduction through a 2-inch brick wall and the temperatures of the two surfaces of the brick wall when the difference of the temperatures remains constantly equal to 100°C . or 180°F . The curve shows that the heat transmitted through the

Let it be assumed that the hotter sooted surface is at 500°C ., absolute, and the colder at 400°C ., absolute; the net heat exchanged between the surfaces can be found by following the vertical line of 500°C . until it cuts the curve, and then from this intersection following the horizontal line to the left or the right scale, according to whether the net heat is desired in metric or English units. The assumed condition gives about three calories. When the temperature of the hotter surface is doubled, the difference between the two surfaces remaining 100°C . or 180°F ., the heat exchanged increases to 26 calories, or nearly nine times as much as in the first case.

The lower curves show the approximately similar relation between two surfaces not sooted. The conditions for both curves must be those of figures 12 and 13.

brick remains constant no matter what the temperatures of the surfaces are, so long as the temperature difference remains equal to 100°C .

The lower horizontal straight line shows the same relation for a brick wall 4 inches thick. The heat passing through the 4-inch

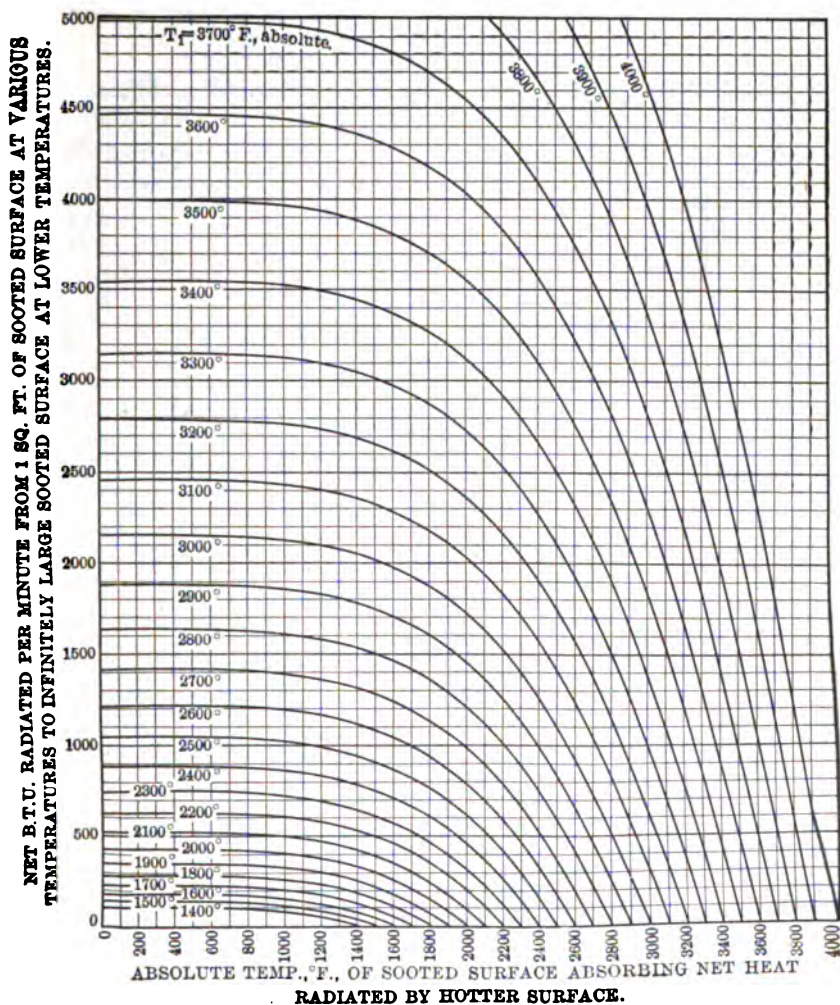


FIGURE 18.—Radiation of heat from a hot surface to a cooler surface; relations expressed in British thermal units, degrees Fahrenheit, and square feet.

wall is one-half of that passing through the 2-inch wall, which, of course, might be expected. The chart shows that the curves of radiation start below the lines of conduction, but as the temperatures increase both of the radiation curves cross the lines of conduction and beyond that rise very rapidly.

APPLICATION OF THE LAWS.

The chart further shows that an air space is more advantageous than a 2-inch brick wall if the temperature of the hotter surface is below 625° C., absolute, or 565° F., ordinary scale, but above that temperature the brick wall is better. With a 4-inch wall the limiting temperature drops down to 500° C., absolute, or 440° F., ordinary

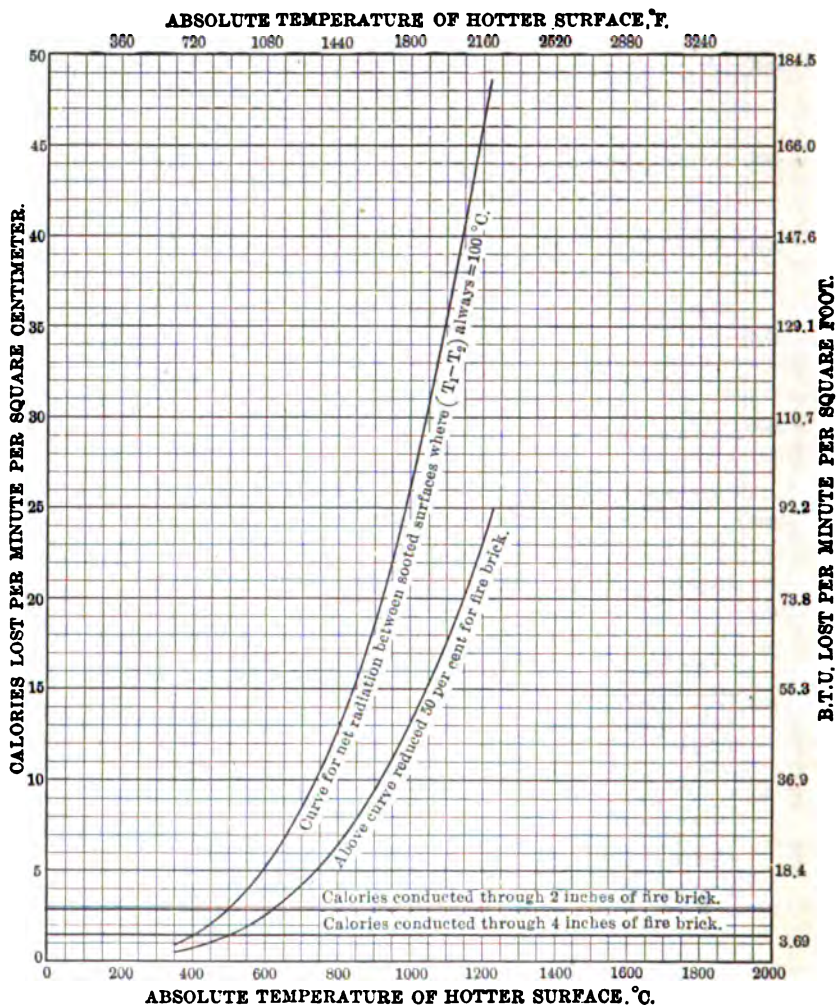


FIGURE 19.—Comparison of laws of heat radiation and heat conduction.

scale. If asbestos, ash, or other better insulating materials were used in place of the brick, this limiting or dividing temperature would fall still lower. These statements hold particularly for the condition of 100° C. temperature difference. There is one lesson which the chart brings out rather emphatically and that is, when

heat at low temperature is to be insulated, use air space; when the heat is at high temperature, as is the case in furnaces, use solids of poor heat conductivity. That the space is less effective at high temperatures than at low ones is known to the makers of "thermos" bottles, who advertise that such bottles keep liquids cold 72 hours and keep liquids hot 24 hours.

PUBLICATIONS ON FUEL TESTING.

The following publications, except those to which a price is affixed, can be obtained free by applying to the Director of the Bureau of Mines, Washington, D. C. The priced publications can be purchased from the Superintendent of Documents, Government Printing Office, Washington, D. C.:

PUBLICATIONS OF THE UNITED STATES GEOLOGICAL SURVEY.

BULLETIN 261. Preliminary report on the operations of the coal-testing plant of the United States Geological Survey at the Louisiana Purchase Exposition, in St. Louis, Mo., 1904; E. W. Parker, J. A. Holmes, M. R. Campbell, committee in charge. 1905. 172 pp. 10 cents.

PROFESSIONAL PAPER 48. Report on the operations of the coal-testing plant of the United States Geological Survey at the Louisiana Purchase Exposition, St. Louis, Mo., 1904; E. W. Parker, J. A. Holmes, M. R. Campbell, committee in charge. 1906. In three parts. 1492 pp., 13 pls. \$1.50.

BULLETIN 290. Preliminary report on the operations of the fuel-testing plant of the United States Geological Survey at St. Louis, Mo., 1905, by J. A. Holmes. 1906. 240 pp. 20 cents.

BULLETIN 323. Experimental work conducted in the chemical laboratory of the United States fuel-testing plant at St. Louis, Mo., January 1, 1905, to July 31, 1906, by N. W. Lord. 1907. 49 pp. 10 cents.

BULLETIN 325. A study of four hundred steaming tests made at the fuel-testing plant, St. Louis, Mo., 1904, 1905, and 1906, by L. P. Breckenridge. 1907. 196 pp. 20 cents.

BULLETIN 332. Report of the United States fuel-testing plant at St. Louis, Mo., January 1, 1906, to June 30, 1907; J. A. Holmes, in charge. 1908. 299 pp. 25 cents.

BULLETIN 334. The burning of coal without smoke in boiler plants; a preliminary report, by D. T. Randall. 1908. 26 pp. 5 cents. (See Bull. 373.)

BULLETIN 336. Washing and coking tests of coal and cupola tests of coke, by Richard Moldenke, A. W. Belden, and G. R. Delamater. 1908. 76 pp. 10 cents.

BULLETIN 339. The purchase of coal under government and commercial specifications on the basis of its heating value, with analyses of coal delivered under government contracts, by D. T. Randall. 1908. 27 pp. 5 cents. (See Bull. 428.)

BULLETIN 343. Binders for coal briquettes, by J. E. Mills. 1908. 56 pp.

BULLETIN 362. Mine sampling and chemical analyses of coals tested at the United States fuel-testing plant, Norfolk, Va., in 1907, by J. S. Burrows. 1908. 23 pp. 5 cents.

BULLETIN 363. Comparative tests of run-of-mine and briquetted coal on locomotives, including torpedo-boat tests and some foreign specifications for briquetted fuel, by W. F. M. Goss. 1908. 57 pp., 4 pls.

BULLETIN 366. Tests of coal and briquettes as fuel for house-heating boilers, by D. T. Randall. 1908. 44 pp., 3 pls.

BULLETIN 367. Significance of drafts in steam-boiler practice, by W. T. Ray and Henry Kreisinger. 1909. 61 pp.

BULLETIN 368. Washing and coking tests of coal at Denver, Colo., by A. W. Belden, G. R. Delamater, and J. W. Groves. 1909. 54 pp., 2 pls.

BULLETIN 373. The smokeless combustion of coal in boiler plants, by D. T. Randall and H. W. Weeks. 1909. 188 pp. 20 cents.

BULLETIN 378. The purchase of coal under government specifications, by J. S. Burrows. 1909. 44 pp. 10 cents. (See Bull. 428.)

BULLETIN 382. The effect of oxygen in coal, by David White. 1909. 78 pp. 3 pls.

BULLETIN 385. Briquetting tests at the United States fuel-testing plant, Norfolk, Va., 1907-8, by C. L. Wright. 1909. 41 pp., 9 pls.

BULLETIN 392. Commercial deductions from comparisons of gasoline and alcohol tests on internal-combustion engines, by R. M. Strong. 1909. 38 pp. 5 cents.

BULLETIN 393. Incidental problems in gas-producer tests, by R. H. Fernald, C. D. Smith, J. K. Clement, and H. A. Grine. 1909. 29 pp. 5 cents.

BULLETIN 402. The utilization of fuel in locomotive practice, by W. F. M. Goss. 1909. 28 pp.

BULLETIN 403. Comparative tests of run-of-mine and briquetted coal on the torpedo boat *Biddle*, by Walter T. Ray and Henry Kreisinger. 1909. 49 pp. 10 cents.

BULLETIN 412. Tests of run-of-mine and briquetted coal in a locomotive boiler, by Walter T. Ray and Henry Kreisinger. 1909. 32 pp.

BULLETIN 416. Recent development of the producer-gas power plant in the United States, by R. H. Fernald. 1909. 82 pp., 2 pls. 15 cents. Reprinted as Bureau of Mines Bulletin 9.

BULLETIN 428. The purchase of coal by the Government under specifications, with analyses of coal delivered for the fiscal year 1908-9, by G. S. Pope. 80 pp. 10 cents. Reprinted as Bureau of Mines Bulletin 11.

PUBLICATIONS OF THE BUREAU OF MINES.

BULLETIN 1. The volatile matter of coal, by H. C. Porter and F. K. Ovitiz. 1910. 56 pp., 1 pl.

BULLETIN 2. North Dakota lignite as a fuel for power plant boilers, by D. T. Randall and Henry Kreisinger. 1910. 42 pp., 1 pl.

BULLETIN 3. The coke industry of the United States as related to the foundry, by Richard Moldenke. 1910. 32 pp.

BULLETIN 4. Features of producer-gas power-plant development in Europe. 1911. 27 pp., 4 pls.

BULLETIN 5. Coking and washing tests of coal at Denver, Colo., July 1, 1908, to June 30, 1909, by A. W. Belden, J. W. Groves, and K. M. Way. 1910. 62 pp.

BULLETIN 6. Coals available for illuminating-gas manufacture, by A. H. White and Perry Barker. 1911.

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**DEPARTMENT OF THE INTERIOR
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THE USE OF PERMISSIBLE EXPLOSIVES

BY

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PREFACE.

A publication entitled "A Primer on Explosives for Coal Miners," which was reprinted as Bureau of Mines Bulletin 17, contains a general account of the properties of the explosives commonly used for blasting coal, a description of the devices used for firing such explosives, and some general directions on the use of explosives. The present bulletin is printed as a supplement to Bulletin 17 and is intended primarily for coal miners, especially for those working in fields where inflammable gas or coal dust may cause disastrous mine explosions. The bulletin is issued by the Bureau of Mines in the hope that it may prove helpful in lessening accidents due to the use of explosives, and in particular may aid in preventing mine explosions, many of which have been caused by the use of improper explosives.

The authors wish to acknowledge the courtesies shown them by various coal-mining companies in the course of visits to mines at which permissible explosives are used. In particular, they wish to thank the officials of the Consolidation Coal Co., Fairmont, W. Va., for cooperation in obtaining photographs to illustrate the methods of using explosives, and the Tennessee Coal, Iron, and Railroad Co., Birmingham, Ala., for the material used for the illustrations showing the arrangement of shot-firing wires in headings and rooms.

THE USE OF PERMISSIBLE EXPLOSIVES.

By J. J. RUTLEDGE and CLARENCE HALL.

INTRODUCTION.

Many of the fatal accidents in coal mines have resulted from the use of explosives for breaking down the coal. These accidents have resulted both from the use of explosives that were not suitable and from the careless or improper use of suitable explosives.

Investigation has shown that one of the commonest causes of disastrous explosions in the coal mines in this country has been a blown-out shot of black blasting powder or dynamite. The dangers that attend the use of these explosives were perceived several years ago, and in consequence the attention of powder manufacturers was directed to the production of explosives that would be less liable to ignite inflammable gas or dust. The manufacturers have been so successful in their efforts that it is now possible to obtain explosives which yield much shorter and quicker flames than black blasting powder or dynamite, and hence are much less dangerous to use in fiery or dusty coal mines.

To find out which of the many grades of short-flame explosives have the qualities that render an explosive most effective and least dangerous for mining coal, the United States Government established a testing station at Pittsburgh, Pa., in 1908. This station is now under the charge of the Federal Bureau of Mines. Before the station was opened a statement of the requirements that the explosives in question would have to satisfy was drawn up and sent to manufacturers. A number of these sent explosives to be tested, and on May 15, 1909, a list of the explosives that had passed the tests was published. Since that date other brands of explosives have been tested and three more lists have been issued. The latest list, published in Miners' Circular 2, Bureau of Mines, contained the names of no less than 71 explosives that had passed the tests prior to January 1, 1911.

These explosives are termed permissible explosives and are thus defined: An explosive is termed a permissible explosive when it is similar in all respects to the sample that passed certain tests by the Federal Bureau of Mines, and when it is used in accordance with the conditions prescribed by the bureau.

The tests that the explosives must pass and the conditions under which they are deemed permissible are given in two publications, Bulletin 17 and Miners' Circular 2, of the Bureau of Mines.

All the explosives named in the published tests of permissible explosives are short-flame explosives; but there are short-flame explosives that have not yet satisfied the requirements of the Bureau of Mines and hence are not permissible explosives.

Permissible explosives are now used to a considerable extent in the coal mines of Pennsylvania, West Virginia, Alabama, Colorado, and to a less extent in those of southern Illinois, Oklahoma, and other States. Their use increased greatly during the last half of the year 1909, the quantity sold in that period being nearly three times as great as that sold in the whole of 1908. This rapid increase is believed to have resulted in part from the publication of facts about these explosives by the United States Government and in part from repeated public demonstrations at the Pittsburgh station which proved that shots of black blasting powder ignited the dust of bituminous coal, but shots of permissible explosives did not.

In order to find out how fast the use of better explosives was increasing, the Government sent inquiries to various makers of explosives in this country. In reply to these inquiries the manufacturers gave their yearly sales of short-flame explosives for the years 1900 to 1909. From these statements two tables have been prepared. The first table shows the quantities sold during each year from 1900 to 1909. The second gives the quantities sold during the first and second halves of the year 1909 and names the different coal fields in which the explosives were sold that year.

Yearly sales of short-flame explosives in the United States.

Year.	Pounds.	Year.	Pounds.
1900.....	1905.....	1,031,300
1901.....	1906.....	1,533,575
1902.....	11,300	1907.....	2,095,244
1903.....	288,661	1908.....	2,108,610
1904.....	608,270	1909.....	8,942,857

* Includes 8,598,027 pounds of permissible explosives.

Quantity of permissible and short-flame explosives sold in different coal fields in 1909.

Coal fields.	From Jan. 1 to June 30.	From July 1 to Dec. 31.	Total.
	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>
Pennsylvania anthracite.....	329,330	564,964	894,294
Northern Appalachian.....	2,132,875	3,843,750	5,976,625
Southern Appalachian.....	618,900	816,050	1,434,950
Eastern interior.....	750	6,205	6,955
Western interior.....	82,500	171,975	254,475
Rocky Mountain.....	12,700	134,000	146,700
Pacific coast.....	29,610	199,248	228,858
	3,206,665	5,736,192	8,942,857

^a Includes 8,598,027 pounds of permissible explosives.

The first table shows that the total production of short-flame explosives in the years 1902 to 1908, inclusive, was 7,676,960 pounds, and that the production in 1909 was 8,942,857 pounds; in other words, the production of these explosives in the single year 1909 exceeded the entire production in the United States up to January 1, 1909, by 1,265,897 pounds.

The second table shows how rapidly the use of short-flame and permissible explosives is increasing in the different coal fields. The table, in a way, also shows how coal mining is being made safer for the miner, because each pound of these short-flame explosives used takes the place of at least a pound, and probably much more than a pound, of the more dangerous black powder. The demand for the permissible explosives is shown by the fact that in 1909 the total sales of such explosives in the different coal fields amounted to 8,598,027 pounds, or more than 96 per cent of the total sales of the short-flame explosives in that year. The returns received from the manufacturers indicate that the sales of permissible explosives for the year 1910 reached 12,000,000 pounds.

Permissible explosives may be arranged in four classes that are based on the nature of the substances used in making the explosives and the proportions used of these substances. These classes are: Ammonium nitrate, nitroglycerin, nitro-starch, and hydrated explosives. Some of the manufacturers of permissible explosives make several grades of each of these classes, each grade being suited for some of the varying conditions that exist in the coal beds of different regions. It is possible by experiment to make a permissible explosive suitable for shooting any coal that is properly undercut or sheared, just as it is possible to get a grade of black blasting powder suitable for blasting the same coal. Manufacturers are trying to make explosives of the "permissible" type that

will meet the requirements in each of the coal fields in this country, that will do any kind of blasting as well as black blasting powder, and that will do it with less danger to the miner. In these efforts the manufacturers have already made good progress.

Most permissible explosives have the general character of "high" explosives, and all are entirely different from black blasting powder in composition and action. They give a shorter flame, explode more quickly, and are much more powerful. To use them properly the miner must allow for their greater quickness and strength.

METHODS OF USING PERMISSIBLE EXPLOSIVES.

The authors of this bulletin visited many coal mines in which permissible explosives were being used, and noted the ways in which the explosives were employed to shoot beds of coal that differed in hardness, in height, and in the character of the joints, partings, etc. They gave careful attention to the ways in which holes were placed and charged and the methods of firing shots.

PLACING AND CHARGING OF SHOT HOLES.

Coal beds vary greatly in thickness and in other features from place to place, so the best position and arrangement of shot holes can not be the same in all coal fields. In some districts, shots of permissible explosives must be placed 2 feet from the roof and ribs in order to give best results, whereas in other districts the shots must be placed 6 inches from the roof and ribs. The "lift" of the hole (the angle the hole makes with a horizontal plane) must also vary in different fields. In some districts the holes are driven nearly flat and in others they make an angle of 30° with the horizontal. However, holes for permissible explosives never require as great a lift as do holes for black blasting powder, and level holes that "cut out" horizontally give better results than holes with considerable lift. Generally all holes are driven parallel to the ribs, but if a face is uneven one drill hole may be driven at an angle with the rib in order to square up the face. The holes should never be drilled on the solid, and should never be driven any farther than the undercutting or shearing extends.

In using permissible explosives the miner should take all the precautions that a careful miner would take in using black powder. The permissible explosives are much stronger, bulk for bulk, than black powder, and they will not give the best results unless they are used with judgment. From their observations in many coal mines, the writers of this bulletin are convinced that permissible explosives work to best advantage and yield the largest proportion of lump coal when the face is undercut properly and then blasted by several successive shots. It is much better and

safer to use a number of small shots than a few large ones. As a rule, increasing the number of shot holes, up to a certain limit, increases the proportion of lump coal. In the best practice, all holes are so drilled as to take every possible advantage of bands or partings in the coal bed, and each shot is fired and the coal is loaded out before the next shot is fired.

Each permissible explosive is sold in cartridges that are always made of a certain diameter and length. Consequently
Size of cartridge. the miner can tell exactly how large the drill hole must be to give good results from a charge of each explosive, and can keep his coal auger at a certain definite gage.

Some permissible explosives are sold in cartridges that are too large to be inserted in the drill hole. If the diameter
Methods of charging holes. of a cartridge is too large the cartridge is usually slit and the explosive is removed and put in a loosely rolled paper case that is just large enough to fit the drill hole. To make a new case in this way takes time and delays the miner in preparing his shots. This procedure is not recommended, because tests at the Pittsburgh testing station show that any change made in the kind or quantity of paper in the case causes a change in the gaseous products of explosion and, in some cases, in the safety qualities of the explosive. If the cartridges supplied have considerably less diameter than the drill hole, some miners slit the cartridges into halves lengthwise and tamp the separate pieces tightly into the hole. This method is followed where the coal requires a quick explosive. If a slower action is needed, it may be had by leaving an air space in the drill hole about the cartridge.

Because of the delay caused by making new cartridges, the miner usually prefers explosives in cartridges that fit the drill hole and can be used just as they are supplied. If the cartridges fit the hole snugly, no time is lost in charging shots, whether miners or shot firers do this work.

Moist clay made into "dummy" cartridges of slightly smaller diameter than the drill holes is employed for stemming
Clay for stemming. at most mines where permissible explosives are used. This clay is either dug outside the mine, brought in by mine cars, and left along the entries at points within easy reach of the miners, or it is dug inside the mine, from the clay bed underlying or overlying the coal, and distributed along the entries.

At some mines the writers saw shots tamped with coal spalls, but the practice is not recommended. Coal cuttings and
Dangerous stemming. fine dust made by the cutting machine or the auger have been used for stemming, but their use is dangerous. Even if they are wet, the relatively cool flame of a permissible explosive may ignite them.

Wooden rods that fit the drill hole snugly should always be employed as tamping bars. Iron or steel rods should never be used, and even copper-tipped bars are not to be recommended.

Tamping bars.

The tightness of the tamping for permissible explosives varies considerably in different fields. Some coals seem to break best if the stemming is tamped tightly; other coals break best if the stemming is packed loosely near the explosive and tamped tightly throughout the rest of the hole. If an explosive is too quick acting for a certain coal, many miners put in a plug of soft, tough clay so as to leave a small air space between the explosive and the stemming. Such an air space lessens the charging density;

Use of air space.

that is, it makes the charge smaller in proportion to the size of the chamber. The effect of the air space is to reduce the shattering action of a quick explosive enough to suit the particular coal being blasted. If an air space is left about the charge, the miner or shot firer must take great care to ram firmly the outer part of the stemming.

At several mines the writers saw that the first or opening shot, on the success of which all the following shots in the face depended, was fired with black blasting powder, because this explosive acts more slowly than the permissible explosives used. In these mines the miners had the option of using either black powder or permissible explosives in the opening shot. To use two explosives in such a way is not good practice. There is too great a chance of the explosives getting mixed; and if different explosives are used in the same drill hole fires are likely to result. Aside from the danger of fires, black powder should not be used in bituminous mines that are dusty or in mines that make any gas (methane) at all, for in such mines shots of black powder may start explosions, no matter how well the shots are placed and charged.

Permissible explosives and black powder at the same working face.

FIRING OF SHOTS.

In those mines that use the permissible explosives in the best and safest ways, shot firers are employed, and the shots are exploded by electric detonators.^a Each shot firer carries with him on his rounds a number of electric detonators and a firing machine. In the interior and far western coal fields it is the rule, and in the majority of the Western States it is required by law that shot firers make their rounds only after the miners have left the mine. In some of the mines in the Appalachian fields (Pennsylvania to Alabama) it is customary to fire the shots during the day shift. There the shot firer's round of the places is so timed that when

Electric shot firing.

^a See pp. 29-32 for a more detailed description of electrical shot firing.

he arrives at each working place the miner has generally prepared the hole, has placed the stemming and the explosive near the drill hole, and is waiting for him. The shot firer examines the face, estimates the quantity of explosive needed, fits the electric detonator to the explosive, and then inserts and tamps the charge. The miner usually helps to load the stemming. As soon as the tamping is done, the detonator legs are properly connected to one end of a piece of double leading wire or cable from 75 to 100 feet long, and the farther end of the cable is carried back to a crosscut or other convenient place out of danger from the shot. Then, after everyone has left the working place, the wires are connected to the firing machine and the shot is fired. This plan, if carefully followed, greatly reduces the dangers of accidents from shot firing during a shift. Generally there is a loss every few days of a foot or two of leading wire, due to the breakage of the ends of the wires, but this loss is too small to be considered.

At the mines where this plan is now in use, the number of shot firers needed is about one for each 200 to 300 tons of run-of-mine coal per day, provided the coal has been properly undercut before shooting.

An advantage of the system of shot firing by electricity is that the operators usually furnish the electric detonators, and the detonators they supply are more apt to be uniformly good than those bought by the miners at different stores. Another advantage is that the operator knows that the detonators are of the right strength—a most essential matter—and can fix the responsibility if poor detonators are used.

In one mine visited, shot holes with extra heavy heels were sometimes loaded with two charges, and these charges were fired in series. For example, in a 6-foot hole with two sticks of permissible explosive at the bottom, half a stick was placed at a point $2\frac{1}{2}$ feet from the mouth, and the remainder of the hole was tightly tamped. The legs from the electric detonators were arranged to connect the charges in series; that is, one after the other. When the shot was fired, the extra charge blew off the heavy heel. This practice is dangerous, because if the outer detonator is defective and does not explode, and the inner one is good and explodes, the outer charge will be blown out of the hole, and its flame may ignite gas or dirt. Wherever shots are so placed

that they depend one on the other for relief, the best practice demands that the single holes should be fired separately in their proper order.

Ordinary fuse has been and still is generally used for firing such shots as the miner himself fires. Fuse and detonator are also used in some mines in which shot firers are employed. Where fuse is employed in firing permissible explosives, the common practice is to load and fire the holes shortly after they

are drilled. Although convenient, the practice is not recommended for use in coal mines in which inflammable gases occur. The chance of an explosive not detonating properly is always greater with fuse than with electric detonators. The chief danger, however, is in the ignition of gas. The naked flame used to light the fuse, or the flame from the fuse itself, may ignite gas at the face and start an explosion.

Recently a device has been placed on the market that is said to prevent the spitting of fuse, and hence the ignition of gas. This device consists of a tube that fits tightly over the outer end of the fuse and contains a composition that is ignited by turning a projecting wire. The device is successful in preventing the spitting of the fuse generally offered for sale in this country, but its value is greatly lessened by the fact that flame frequently bursts from the side of the fuse.

In some mines there is a tendency to economize by using short lengths of fuse. This is a bad and dangerous practice, for it is apt to result in a shot going off too soon.

Since electric detonators cost about as much as good fuse of the same length and are much safer, they should always be used in gaseous and dusty coal mines.

Fuse may cause misfires because of kinks or cuts, or because the gunpowder core is missing in places, or because the detonator, from not having been securely crimped, works loose from the fuse.

If the proper precautions are not taken, shots of permissible explosives may misfire. Such failures may be caused in one or more of the five following ways:

1. Imperfect detonation; generally the result of using a detonator that is too weak.
2. A displaced detonator.
3. The use of frozen or damaged explosive.
4. A short circuit caused by an imperfectly connected wire, or by the leading wire or the legs of the electric detonator having lost part of their insulation.
5. Defects in the battery or firing machine.

A statement of the causes of misfires indicates how misfires may be prevented. The detonators should be fresh and strong enough to explode the charge completely; they should be properly placed in the cartridge; the leading wires and the detonator legs should be properly insulated, and the battery should be in good condition. Leading wires and detonators should be carefully examined before each shot, and any bare places that are found should be properly covered. All frozen explosives should be thoroughly thawed before use.

At several mines in the eastern districts of the United States, the **Shot firing by miners.** miners fire the shots of permissible explosives with fuse and detonator. The shots are generally fired either before the shift or in the early part of the shift. An examination of the results at the mines visited showed that this practice does not give as good results as having the firing done by special men, because of the miner's tendency to use more explosive than is needed to bring down the coal. This overcharging of shots increases the percentage of fine coal. Another disadvantage of such shot firing by miners is that the shots are not fired systematically. Still another and much more serious objection, particularly in gaseous mines, is the increased danger of gas being ignited, either in lighting the fuse or by the burning fuse itself. Many gas explosions have started from shots

Danger of the practice. fired by miners. In addition, the dangers that attend handling all explosives are much increased by allowing miners to fire shots, because the explosives and detonators may be handled by men who are not familiar with the properties of the explosives or who disregard the risks in their use.

The firing of permissible explosives in the chief coal-mining districts **Employment of shot firers.** has been done by special men whose sole duty is to fire shots. The shot firer carries either a shot-firing lamp, a magneto machine, or a dry-cell battery with a safety-contact button. He goes from one working place to another loading and tamping the shots and, as soon as they are ready, firing them. No shot is supposed to be fired unless the working place is properly timbered and the shot firer is satisfied that the shot is properly placed. As the shots are fired in order, as far as is possible, the miner knows when to expect the shot firer and usually has his shots ready for loading and tamping as soon as the latter arrives.

Compared with the plan of allowing miners to fire shots, the employment of a shot firer leads to larger yields of lump coal, **Advantages of employing shot firers.** regulates better the quantity of explosive used, results in more systematic work, and maintains stricter discipline. Another advantage is that the plan prevents one miner loading out all his coal before his neighbor, and thus assures the driver having a uniform tonnage to haul during the entire shift.

During recent years the mines that have the best discipline among those situated in districts where there is considerable **Shot firing by entry bosses.** gas, or where the roof is poor and much timbering is required, have been employing entry, district, or "face" bosses. These men are picked miners of much experience. Each entry boss patrols the working places in his charge to see that they are properly timbered and that the miner is always working under safe conditions. Some companies have found the services of these men of great value and there can be no doubt that the general employment of entry bosses

would greatly lessen the number of accidents. If permissible explosives are employed and a mine is properly districted, these entry, district, or "face" bosses may also perform the duties of shot firer. The tamping and firing takes little time and since each boss has to visit each working place anyway, he can easily perform this small additional duty. Some mines in West Virginia and some in Pennsylvania have found this arrangement most satisfactory.

It is extremely difficult to get correct information as to ignitions of gas after shots of permissible explosives. Shot firers and miners are apt to fail to report such ignitions, and when they refer to them later they do not recall the exact facts in each case. From what the writers have seen while following shot firers on their rounds, it seems extremely probable that by far the larger number of the ignitions laid to permissible explosives are really caused by bringing an open lamp to the face immediately after shooting, and holding it close to the face to see the results of the shot.

If the coal is undercut and the holes properly charged, the shots do not scatter the coal, but merely drop it a few inches and open cracks and crevices at the roof and along the bedding planes; consequently the gases liberated from the coal and the gases from the explosive itself do not escape as freely as when, with excessive charges of black powder, the coal is blown away from the face and shattered. The inflammable gases from properly placed shots of permissible explosives diffuse slowly, and are easily ignited by an open flame held near the face immediately after shooting. The risk of ignition is just about the same as it would be if black blasting powder were used in smaller charges than are customary, and the coal were not blown from the bed but merely dropped on the floor. Some experiments made in a mine in northern West Virginia, in which open lights were used and little or no gas was found under ordinary conditions, showed the risk to be about equal with the two explosives. When open lights were brought to faces freshly blasted by permissible explosives and touched to cracks in the coal, gas was ignited in nearly every trial. When black powder was used and the coal was not scattered but merely dropped on the floor, the gas in the cracks of the face was ignited in just about the same proportion of trials.

If the interval between firing a shot of a permissible explosive and bringing a naked flame to the face is at least five minutes, there should be little danger of the naked flame igniting gas.

EFFECT OF PERMISSIBLE EXPLOSIVES ON SIZE OF COAL.

Permissible explosives have come into use so recently that it is not easy to get reliable figures showing the increased proportion of fine coal they make as compared with black blasting powder. The estimates of this increase given by mine superintendents run from no increase to 10 per cent. Some superintendents maintain that although smaller lumps of coal may be made by using permissible explosives, yet changing from black blasting powder to these explosives does not increase the proportion of fine coal. Some persons state that the lumps of coal produced by using permissible explosives are not so easily broken up during transportation or exposure to the air as are those made by using black blasting powder, whereas other persons maintain the reverse. However, if the coal is undercut or sheared and the blasting is done with judgment, the permissible explosives make as good coal as black blasting powder and at approximately the same cost.

When a mine makes the change from black powder to a permissible explosive, there may be a slight increase in the proportion of fine coal made; but a considerable part of this increase must be laid to the miners' lack of experience with the new explosive. Miners must become accustomed to the use of a new grade of black blasting powder before they can get good results with it, and each miner must learn by experience how best to shoot any coal bed that is new to him. In like manner, a miner must become accustomed to using permissible explosives, for these explosives are entirely different from black powder and must be used in a different way.

From what the writers have seen in mines that use permissible explosives, they have come to believe that miners are apt to try to blast a thick coal bed with only one or two shots instead of using several shots, as they ought to. If such a bed were not undermined, either by a puncher or a chain machine, and the coal were being shot off the solid with black blasting powder, a miner would never think of trying to blast the bed in one bench, but would break the coal down by several shots, each shot depending for its success on the preceding one. The miner should use permissible explosives with equal judgment; if he does he will find that the proportion of lump coal produced compares very favorably with what he would get by the use of black powder.

COST OF MINING WITH PERMISSIBLE EXPLOSIVES.

In a certain coal field, black blasting powder, which was said to cost the operator \$0.95 or more per keg of 25 pounds, was sold to the miner at an agreed price of \$1.75 per keg. In some coal mines, especially those in which coal is shot off the solid and large quantities of powder are used daily, the profit on the powder is considerable. Consequently, operators do not care to change from black blasting powder and lose the profit on it. A way of adjusting this difference when a change is made to permissible explosives is to find out the cost of powder for each ton of coal produced and to charge the permissible explosives to the miner at this rate, allowing for the difference in quantity needed to do the same work.

The price that the operator pays for the permissible explosives he furnishes the miner varies from .12 to 14 cents per pound. It is said that as a rule no profit has been made by the operator from selling these explosives, because the price charged the miner has been just enough to cover the wholesale price the operator paid and the expense of handling.

The miner usually has to pay 0.5 cent per foot for fuse and 0.5 cent to 0.75 cent each for detonators. Where electric detonators are used it has been the practice for the operator to furnish them to the miner free. These detonators cost from 3 to 5 cents each, the price depending on the strength of the detonating composition and the length of the wires. One company that worked a coal bed about 7 feet thick and used electric detonators to fire the shots, reported that the cost of detonators was 3.5 cents for each shot. The cost for permissible explosives at the mine was 1.75 to 2 cents for each ton of run-of-mine coal produced. The daily output was 800 tons and the company paid each of the four shot firers it employed \$2.75 per shift.

According to information received by Government officials from other districts where the cost of using permissible explosives has been carefully kept, there is little if any difference in the cost of blasting a ton of coal, whether permissible explosives or black powder are used. It is true that the comparison is based on different conditions; the figures for black powder were for mines where miners were loading and firing charges in a wasteful manner, whereas those for permissible explosives were from mines where shot firers were employed and the explosives were used in an intelligent way.

Records kept by another company show that the use of permissible explosives slightly increases the blasting cost per ton of coal produced, but the comparison with black blasting powder was made

when the loading and firing of powder and of permissible explosives were done by shot firers.

In one mine that used nothing but black blasting powder and employed shot firers during the year 1909, the average cost of blasting, including the cost of squibs and the wages of the shot firers, was 3.15 cents for each ton of coal produced. The cost varied from 5.4 cents in March to 2.7 cents in August. The cost of shot firing averaged 1.1 cents per ton of coal. In the same district, in another mine, which used permissible explosives during 1909, the average cost of blasting, including the cost of electric detonators and the wages of the shot firers, was 3.3 cents per ton and the average cost of shot firing was 1.22 cents per ton.

Before permissible explosives leave the factory they are put in cartridges having a heavy paper wrapper that is made almost water and weather proof by a coating of paraffin or a similar substance. If the cartridges are carelessly handled, the paraffin coating may be broken; air and moisture can then act on the explosive so that it will lose strength and finally have no explosive power at all. Long exposure to the moist air of the mines or contact with water will also weaken and spoil the explosive, even though the paper cover is not broken.

**Improper handling
and care of permis-
sible explosives.**

MISUSE OF PERMISSIBLE EXPLOSIVES.

At the mines they visited the writers saw a number of instances of the misuse of permissible explosives. The most common form of misuse was the overloading of holes. This happened mostly at mines where miners fired their own shots and regular shot firers were not employed. A miner tries to bring down the coal completely without doing any more pick work than he has to, so he uses more explosive than is necessary. A shot firer, on the other hand, generally tries to get the maximum proportion of lump coal, and so uses only enough explosive to bring down the coal. For this reason a better quality of coal is had by employing shot firers than by allowing miners to charge and fire shots.

Excessive charges.

Another bad practice, said to exist at some mines using shot firers, is the insertion of small quantities of explosives in the back of holes before the arrival of the shot firer and without his knowledge. The miner may do this to hide the fact that the hole has been drilled into the solid and beyond the mining, or he may do it to insure the charge being large enough to satisfy him. However, the practice is most dangerous, and can not be condemned too strongly. At least one instance is on record of a miner putting dynamite into a drill hole that was to be charged with a permissible explosive. In this instance, as was to be expected, the shot gave a dangerously long flame, and an ignition and a fire resulted.

Hidden charges.

The offense that miners commit most often is to bring a naked lamp in contact with the gases liberated or given off by the explosion of permissible explosives (see p. 23). Owing to the fact that these explosives when properly used make little or no smoke, the miner is tempted to return to the face immediately after a shot; and usually he does so, especially if he smells no offensive fumes. Then his naked lamp may ignite the gases from the explosive and those liberated from the coal. Tests in certain coal mines have shown (see p. 23) that it is possible to ignite the gases from a crevice in the face after a shot of either a permissible explosive or black blasting powder. Analyses of the crevice gases seem to prove that most of the inflammable gases found after shots come from the coal. A coal that contains much volatile matter may easily take fire from the ignition of crevice gas, and such face fires after they get a good start are very hard to put out.

ADVANTAGES OF PERMISSIBLE EXPLOSIVES.

One of the most important advantages to be gained by using permissible explosives is the exemption from mine fires. If the explosives are properly employed, ignitions of gas or coal seldom happen.

In many mines working a high volatile coal that gives off considerable gas and produces large quantities of fine dust it has been necessary to employ, with every two shot firers, a miner or "fire runner" to follow and put out any fires caused by shots. This procedure was discontinued when permissible explosives were adopted. However, it is good practice to examine the face after a blast, no matter what explosive is used.

Another important advantage is lessened damage to the roof. In blasting coal that has been undercut it is not necessary to give the drill holes as much lift for permissible explosives as for black powder. The holes for the latter, in order to shoot the coal to best advantage, must usually have considerable "lift"; as a result they are often driven into the roof coal or slate, which is so shattered by the blast that close timbering is necessary. But permissible explosives seem to throw down the coal well and to give a good proportion of lump coal if the holes are almost horizontal. Fired in such holes the permissible explosives appear to work out along the joints and the bedding planes of the coal instead of striking up into the roof or slate—provided, of course, that in placing the holes proper advantage was taken of slate bands or partings in the coal bed. Consequently if head coal is to be left for a roof, these explosives enable the miner to keep this roof smooth, firm, and unbroken, so that less timber is required to support it. In one mine

that was leaving top coal in place, the change from black powder to permissible explosives did away with the need of employing a regular timberman.

Charges of permissible explosives can be and generally should be placed 1 or 2 feet from the ribs and will still bring down the coal. In some coal fields the ribs are cut much cleaner by such shots of permissible explosives than they can be cut by shots of black powder. In one mine the replacing of black powder by a permissible explosive increased the yield for each cut $1\frac{1}{2}$ tons because of the trimming effect of the permissible explosive.

On the other hand, if a hole for a permissible explosive is drilled too deep on the solid, the explosive uses up its strength in making a cavity at the back of the hole without breaking down the solid coal. If charges of black blasting powder are fired in such deep holes, blown-out shots often result, but with permissible explosives blown-out shots are rare and are not accompanied by flame.

If rightly used, permissible explosives do not scatter the coal over the gob, where it is lost, nor do they displace props, as frequently results from the use of black powder. When coal properly undermined is blasted with permissible explosives, it is not thrown out nor much displaced, but is merely dropped in its bed, although it is generally broken up enough to be easily loaded.

Another advantage of permissible explosives, when properly used, is that the shots do not give as heavy shocks to the mine air as do shots of black blasting powder. Consequently, brattice, doors, and stoppings are not subjected to air blasts as strong as those often felt when black powder is used. Still another advantage is that it becomes easy to get men to act as shot firers, because there are no windy shots and little danger of dust explosions.

The permissible explosives are cleaner and much more convenient to use and handle than black powder; moreover, they can be prepared and fired much more quickly. If rightly used, they do not scatter coal nor dislodge timbers, and thus they make the working places safer; moreover, they greatly reduce the danger of starting explosions. Permissible explosives also save time, because they do not make smoke and the miner can more readily examine the roof after a blast.

A charge of a permissible explosive takes up only about one-half the space that a charge of black blasting powder heavy enough to do the same work would occupy; hence for the same depth of hole much more stemming can be used with a permissible explosive than with

the black blasting powder and there is a reduction to a minimum of tendency to blow out stemming. This is another reason why the use of permissible explosives tends to do away with blown-out shots, and thus greatly increases the safety of every person in a coal mine.

Still another advantage connected with the use of some permissible explosives is the ease with which they can be used in wet or damp holes. In shooting off the solid in thick coal beds some bottom holes may be so wet that they can not be fired with black powder. In one mine that was visited, permissible explosives were successfully used to shoot such wet bottom holes.

In blasting coals that contain much volatile matter or gas the use of permissible explosives is absolutely necessary. In such mines black powder or dynamite may ignite bodies of fire damp in the advance working places or strong feeders of gas at the face. Permissible explosives give such short, quick flames that if the coal is undercut and the holes are properly placed and charged, the chance of igniting gas is reduced to a minimum.

In two important and widely separated coal-mining districts, southern Illinois and Oklahoma, permissible explosives are used to some extent in driving entries in beds that give off considerable gas and in which the coal is so brittle that it makes much fine dust. The permissible explosives are used in the entries and in some mines in the rooms as well. Before the permissible explosives were tried, the coal was set on fire nearly every day through the ignition of gas feeders, but since the permissible explosives have been used no such fires have been reported.

Permissible explosives are used in several districts to blast coal off the solid. Such use is reported to be successful in certain mines in Alabama, but as a rule does not give nearly as good results as when the coal has been undermined. The coal is usually more broken up than it would be if undercut before shooting, and the necessarily larger charges of explosives tend to weaken the roof and to increase the danger of roof falls.

One of the most important advantages of the use of permissible explosives, if properly fired in suitably prepared drill holes, is the absence of smoke. If just enough of the permissible explosive is used, little, if any, smoke is produced. This feature is especially important in mines where the roof is bad. The ventilating current is not fouled by large quantities of dense smoke, as is the case when black powder is employed, and slips or cracks in the roof can be more easily and quickly seen. However, the charge must be properly fired; otherwise noxious gases may be given off and these may have disagreeable or even dangerous effects upon the miner.

Miners frequently complain of the fumes given off from shots of permissible explosives, but in the many mines using these explosives that were visited by the writers of this bulletin fumes seemed to give little or no trouble. In none of the places examined did the fumes from blasts cause discomfort. If the ventilating current is brought directly up to the working face, no discomfort should be caused by shots of permissible explosives, if these explosives are properly used. This advantage, however, leads to a risk in their use, since the miner is tempted to go back to the face as soon as a shot is fired. This is a very dangerous thing to do, no matter what kind of an explosive is used, and should be strictly forbidden.

If the coal is properly undercut or sheared and a charge of 1½ pounds to a hole has not been exceeded, the quantity of poisonous gases produced by the permissible explosive itself is so small that it is of little, if any, consequence. It is easy to understand why the gases should not be troublesome if one remembers that the volume of gases produced by firing 1½ pounds of an explosive is soon diluted by several thousand cubic feet of air.

Black blasting powder containing Chile saltpeter, and permissible explosives of each of the four classes that have been mentioned (p. 9) were tested by firing shots in coal mines. Electric detonators strong enough to cause complete detonation were used. The charges of permissible explosives varied from 13 to 24 ounces. Samples of air were taken near the face before each shot, and at the same place at intervals of 20 seconds, 2 minutes, and 4 minutes after the shot. To ascertain the quantity of harmful gases that remained in badly ventilated places the experiments were tried in working places ahead of the air, generally in blind entries. None of the samples of the mine air taken immediately after the blasts of the different explosives was found to contain more than a trace of any poisonous gas.

However, although the samples of air collected immediately after blasts of the different explosives did not contain measurable quantities of poisonous gas, yet in all the mine tests of explosives, samples of the gases taken immediately after blasting from the crevices made in the coal by the shot contained large quantities of poisonous or inflammable gas. Because these gases escape from the face slowly, a miner should never return at once to the face after a blast, but should always wait at least five minutes. It is known that the gases continue to come from the crevices for several minutes after a shot. They mix with and are diluted by the air of the working place. Analyses of crevice samples also show that in non-gaseous bituminous coal mines 8 to 27 per cent of inflammable gas, largely methane (CH_4), is formed with each shot of an explosive; this

gas is believed to come from the pores of the coal, whence it is forced or drawn by the explosion of the charge.

Analyses of samples of air collected after shots in mines also show that more of that poisonous gas, carbon monoxide, or white damp, is formed in blasting coal than is formed by exploding equal charges of the same explosives in the pressure gage used to test explosives at the Pittsburgh station. Part of the extra carbon monoxide is believed to come from the ignition by the explosive of fine coal left in the drill hole and part from the coal itself. Explosives that produced no carbon monoxide when exploded in the pressure gage always produced some carbon monoxide in the tests in coal mines, and those that yielded a relatively high percentage of carbon monoxide in the tests in the pressure gage gave a correspondingly large yield in the mine tests.

A typical analysis of the gases collected from a crevice at the face 20 seconds after a shot of black blasting powder containing Chile saltpeter is given below; allowance is made for the air mixed with the gases and for the same percentage of methane that was in the gases resulting from the explosion of the powder in the pressure gage.

<i>Gases from a shot of black blasting powder.</i>		Per cent.
Carbon dioxide (CO ₂).....		19.2
Carbon monoxide (CO).....		28.1
Oxygen (O ₂).....		.0
Hydrogen (H ₂).....		10.0
Methane (CH ₄).....		.6
Hydrogen sulphide (H ₂ S).....		6.8
Nitrogen (N ₂).....		35.3

The analysis shows that the poisonous gases (carbon monoxide and hydrogen sulphide) generated by this black blasting powder form 34.9 per cent of the whole volume of the gases produced.

A typical analysis of the gases, taken under the conditions above referred to, from a permissible explosive that gave in mine tests the largest proportion of poisonous gases, is as follows:

<i>Gases from a shot of a permissible explosive.</i>		Per cent.
Carbon dioxide (CO ₂).....		26.7
Carbon monoxide (CO).....		36.4
Oxygen (O ₂).....		.0
Hydrogen (H ₂).....		14.2
Methane (CH ₄).....		1.4
Hydrogen sulphide (H ₂ S).....		.0
Nitrogen (N ₂).....		21.3

A typical analysis of the gaseous products of a permissible explosive that gave the smallest proportion of harmful gases in the field tests and did not give any carbon monoxide in the gage tests is as follows:

<i>Gases from a shot of a permissible explosive.</i>		Per cent.
Carbon dioxide (CO ₂).....		18.0
Carbon monoxide (CO).....		6.4
Oxygen (O ₂).....		.0
Hydrogen (H ₂).....		8.2
Methane (CH ₄).....		.5
Hydrogen sulphide (H ₂ S).....		.0
Nitrogen (N ₂).....		66.9

Many samples of gas were taken from crevices made in the coal by shots of different explosives, and these samples were analyzed. The analyses show (after allowance is made for the air in the sample and for the proportion of methane found by tests in the pressure gage) that the gas from crevices after blasts with various permissible explosives contained, on an average, 19.5 per cent of carbon monoxide.

The results of these mine experiments indicate that the gases from the explosion of any of the explosives tested are mixed with the gases that come from the coal itself and that the mixtures formed are extremely poisonous and inflammable. Since these gas mixtures are inflammable, it is easy for a miner to ignite them by holding an open flame to a crevice just after a shot. To ignite crevice gas in this way, however, is a most dangerous practice and should be strictly forbidden.

The experiments in mines showed also that four minutes after a shot the greater part of the crevice gases had passed into the mine air and were so diluted that measurable quantities of them could not be found in the samples of mine air collected.

All of the mine tests just mentioned were made in the Pittsburgh district. The coal had been properly undercut and only enough explosive to do the work was used for each shot. If the charges of permissible explosives are heavier than they need be, it is reasonable to suppose that harmful gases in quantities sufficient to be injurious to the health of the men employed may be given off. The remedy, of course, is not to stop using permissible explosives, but to use smaller charges.

The facts gathered by the writers make it clear that if all coal mines in this country would stop using black blasting powder and would use permissible explosives instead, the miners would find their work pleasanter and safer. Of course, the explosives would have to be used properly and would have to be fired so that their detonation would be complete. In other words, the coal would have to be undercut and the shots charged and

Gases from shots are poisonous and inflammable.

Permissible explosives make mining pleasanter and safer.

fired by shot firers who use electric detonators. Unless they are properly used permissible explosives will not give the best results and may yield dangerous quantities of noxious or inflammable gases.

EXAMPLES OF GOOD PRACTICE IN THE USE OF PERMISSIBLE EXPLOSIVES.

The writers, in their examination of mines that use permissible explosives, noted various features of practice. Some features that seemed particularly good will be described here.

In one large mine the headings were driven 10 feet wide and the rooms 20 feet wide, leaving room pillars 40 feet thick. **Driving headings.** The coal was strong; the bed was about 8 feet thick and had a fairly good shale roof. In most places 8 to 12 inches of top or head coal was left to support this roof.

The coal in headings and rooms was undermined by chain breast machines. The rooms were six runs wide and the headings were three

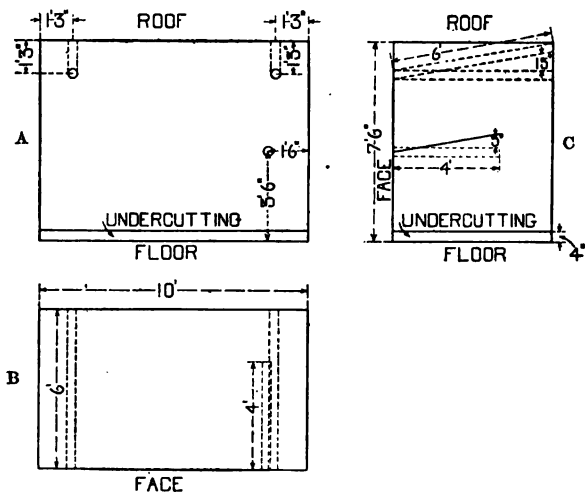


FIGURE 1.—A proper arrangement of holes in a heading with coal undercut. A, front view; B, plan; C, side view.

runs wide. The undercut was about 4 inches high and 6 to 7 feet deep.

In most places the headings were shot by three holes, charged with a permissible explosive of the ammonium-nitrate class. The holes were placed in the following manner:

A "block" hole 3 to 4½ feet deep was started about 3 feet from the floor and driven parallel to and but a few inches from the right rib. It was usually pointed downward at an angle of 3° to 5° with the horizontal.



A. FACE OF A 10-FOOT HEADING WITH THE "BLOCK" HOLE AND THE RIGHT RIB HOLE DRILLED.



B. FACE OF A 10-FOOT HEADING AFTER THE "BLOCK" HOLE WAS FIRED.



A. FACE OF A 10-FOOT HEADING BEFORE FIRING OF RIB HOLES. COAL FROM "BLOCK" HOLE LOADED OUT.



B. FACE OF A 10-FOOT HEADING AFTER BOTH RIB HOLES WERE FIRED.

Two rib holes were driven; one was 6 feet in depth and a few inches from and parallel to the right rib; the other was 6 feet in depth and a few inches from and parallel to the left rib. These two holes were placed like the right rib hole shown in Plate I, *A*; they were started 6 to 18 inches from the upper corner and were pointed upward at an angle of 10° to 15° with the horizontal. Plate I, *A*, shows a view of the face of a heading 10 feet wide with the "block" hole and the right rib hole drilled.

The "block" hole was fired first and, as a rule, yielded a much smaller quantity of lump coal than the other holes. (See Pl. I, *B*.) The usual charge was one and one-half to two sticks (0.75 to 1 pound) of the permissible explosive. When the coal from the "block" shot was loaded out (see Pl. II, *A*), the rib shot over it was fired. An ordinary charge for the rib holes was from one and one-half to two and

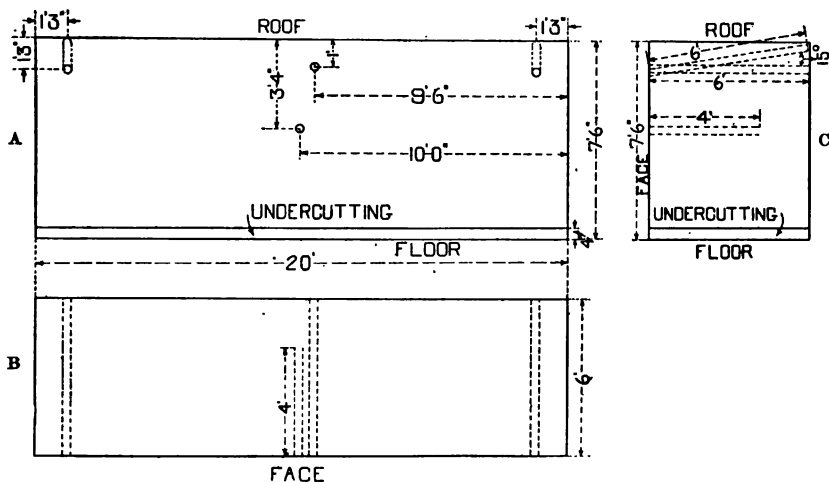


FIGURE 2.—A proper arrangement of holes in room with coal undercut. A, front view; B, plan; C, side view.

one-half sticks. Immediately after this shot and before the coal from the first rib shot was loaded out the other rib shot was fired. Plate II, *B*, shows the results of firing the rib shots. Figure 1 shows the arrangement of shot holes in headings.

The rooms were 20 feet wide and were usually widened out on one side; the coal was undercut to a depth of 6 feet.

Driving rooms.

Four holes (Pl. III, *A*, and fig. 2) were used to shoot the rooms: one "block" hole, one "breaker," and two rib holes. Plate III, *A*, shows the coal face and arrangement of drill holes and figure 2 shows the lengths and angles of the holes.

The block hole was drilled to a depth of 3 to 4 feet either horizontally or pointing downward at an angle of 3° to 5° . This hole was started at about the center of the room and 4 to $4\frac{1}{2}$ feet above the

floor; it was driven parallel to the ribs. The charge, two to three sticks of explosive, was fired after just enough of the "bugdust" made by the mining machine had been loaded out to clear the shot.

The breaker hole was 6 feet in depth. It usually started about 2 feet above the block hole and 8 to 10 inches from the roof, extended upward at an angle of 10° to 15° , but parallel to the ribs, and ended at the roof coal, 8 to 10 inches of which was left in place. The breaker shot was not fired until the coal yielded by the block shot had been entirely loaded out. Generally one and one-half to two and one-half sticks sufficed for the breaker shot, and as the coal was left free by the removal of coal brought down by the block shot the coal yielded by the breaker was of good quality. Plate III, *B*, shows the face after the coal from the block shot had been loaded out and previous to firing the breaker shot. Plate IV, *A*, shows the shot firer and the miner charging the breaker shot.

Sometimes a little shearing was done by the miner just after the coal from the breaker shot had been loaded out and previous to firing the rib shots. This shearing freed the rib shots which, charged with two to two and one-half sticks each, were fired one immediately after the other. These shots usually yielded the best coal from the face, as they were entirely free. Plate IV, *B*, shows the face before the rib shots were fired, and Plate V, *A*, shows the face after firing.

If the block and breaker shots were not exactly in the center of the room or not parallel to the ribs, one of the rib shots was usually smaller than the other. The firing was done entirely by shot firers who patrolled the working places and fired the shots, at the request of the miner, by means of magneto firing machines. No shot was fired until the working place was properly timbered, and no holes were fired that were driven any distance on the solid. A coil of double No. 18 leading wire 75 to 100 feet long was used to carry the electric current from the battery to the face. Electric detonators with 8-foot legs were used to detonate the charge. They cost 4 cents each. One shot firer was employed for approximately each 300 tons of daily output. The miners worked singly in rooms and headings. The company hired the shot firers and furnished the electric detonators. The miners furnished the explosives, which cost them 14 cents per pound. The shot firer generally tamped and fired each shot in three to four minutes' time.

In the mine cited on page 26, moist yellow clay was used for stemming. This clay was obtained outside the mine, brought into the workings on mine cars, and placed at convenient points along the headings. Another mine using permissible explosives employed the fire clay taken from the floor in the mine. In both mines the clay was formed into "dummies," or clay cartridges, by wrapping paper around a cartridge stick and filling the

Stemming.



A. COAL FACE AND ARRANGEMENT OF DRILL HOLES IN A 20-FOOT ROOM, ONE "BLOCK" HOLE, ONE "BREAKER," AND TWO RIB HOLES.



B. FACE OF A 20-FOOT ROOM BEFORE CHARGING OF "BREAKER" SHOT. COAL FROM "BLOCK" HOLE LOADED OUT.



A. SHOT FIRER AND MINER CHARGING "BREAKER" HOLE IN ROOM.



B. FACE OF ROOM BEFORE RIB HOLES WERE FIRED. COAL FROM "BREAKER" HOLE LOADED OUT.



A. FACE IN 20-FOOT ROOM AFTER FIRING OF BOTH RIB SHOTS.



B. MODEL, NEAR MINE MOUTH, OF THE ARRANGEMENT OF SHOT-FIRING WIRES IN A MINE.

case with moist clay. These dummy cartridges were 10 to 12 inches in length and about $1\frac{1}{4}$ inches in diameter. It was customary for the miner to have the explosive and "dummies" ready at the working face when the shot firer made his round of the places.

A round wooden stick, $1\frac{1}{2}$ inches in diameter and 7 feet in length, was used to tamp the powder and stemming.

The drill holes varied in diameter from $1\frac{1}{4}$ to $2\frac{1}{4}$ inches. They were drilled to the depth of the mining and no farther.

Size and position of drill holes. The block and rib holes were driven horizontally whenever they could be, because the shots in the flat holes "cut out" horizontally, yielded better lump coal, and did not "cut up" into the roof coal as much as did those in inclined holes.

ELECTRIC FIRING OF SHOTS.

As a result of efforts to make the firing of shots in coal mines safer, systems of firing by electricity have been developed. These systems, in the best practice, require that all men shall be out of the mine before any shots are fired. The essential features of a good electrical firing system are as follows:

The mine is wired with rubber-covered wires laid in parallel in order that one defective shot may not prevent others from being fired. On the main entries No. 6 rubber-covered wire is used, on the cross entries No. 10 or No. 12, and in the rooms No. 14 wire. These wires are hung from props or from plugs set in the roof. At the mouth of each cross entry there is placed a locked box in which there is a single-pole double-throw switch that is arranged to open downward to avoid accidental contact by falling. A similar, though smaller, switch is in some mines placed at the mouth of each room, or else the room wire is connected to the wire on the cross entry without the use of a switch. At the mouth of the mine there is a shot-firing cabin in which a 100-ampere, two-pole, single-throw switch is placed. There is a similar switch of the same size in the power house. This plan insures that there shall always be at least three open switches between the mine and the power house when the shot is being connected up.

Arrangement of wires.

Figure 3 illustrates the arrangement of shot-firing wires in a mine room. Plate V, *B*, shows a model of the same wiring placed outside the mine.

The operation of the system is as follows: Every employee in the mine is given a brass check with his number stamped upon it. Upon entering the mine he hangs his check upon a check board in the shot-firing cabin. This board has painted upon it numbers which correspond to the numbers on the checks, and each check is hung over the proper number. No one is allowed to

Operation of system.

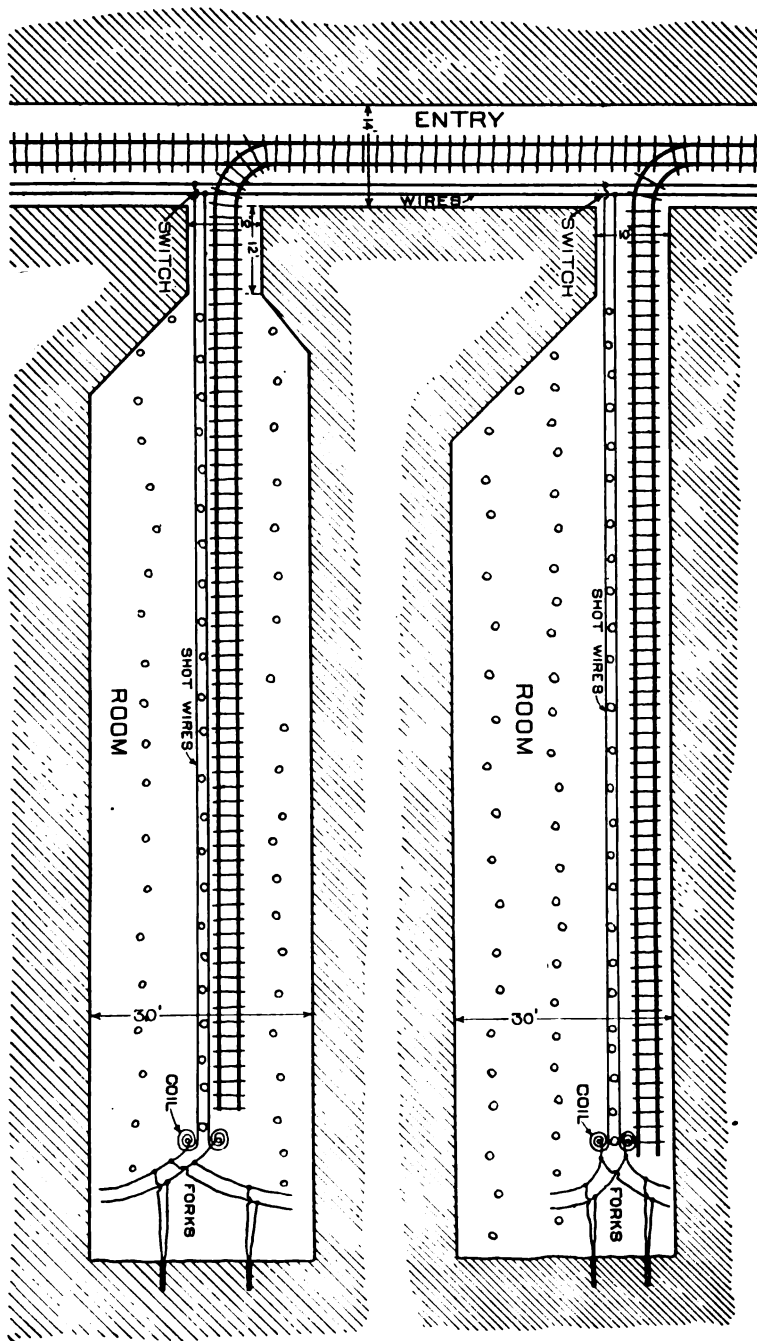


FIGURE 3.—A proper arrangement of shot-firing wires in a mine room.

enter the mine unless he has previously hung up his check on the check board.

Usually the miners are allowed a certain maximum number of shots for each working place—generally three. The coal is undercut in most places, and permissible explosives are used. Special shot firers go through the mine during the day examining the shots as they are being prepared by the miner and distributing the electric detonators. In some instances the miner tamps the shots himself; in others the shots are tamped by the special shot firers. If the miner tamps the shots, he connects the detonator legs sticking out of the shot hole to the No. 14 wire, which extends up the roadway of the room, notes that the connection at the room mouth is made, and then leaves the mine. In connecting the detonator legs to the No. 14 wire care is taken to strip the insulation off the lead wires and to wrap the legs around the No. 14 wire, not looping them, as is done in some mines.

After all the miners have left the mine the shot firer goes through each entry and notes that the switch at the mouth of each room is closed or that the connection is made at the room mouth if the switch is not used. At the mouth of each cross entry he unlocks the switch box, throws in the switch, and locks the box. He then proceeds to the mouth of the mine, enters the shot-firing cabin, and notes that all the checks have been taken off the board, which signifies that the check owners have left the mine. If any checks are still remaining on the board, he searches the mine until the men represented by the checks are found, or if any miner has left the mine without taking his check with him this fact is determined. Usually there is a fine provided for failure to leave or take away a check at the proper time.

When the shot firer has ascertained that all the checks have been taken from the board, he then goes to the power house, where there is a switch provided, and generally an electric generator giving direct current of about 500 volts, to fire the shots. The shot firer instructs the man in the power house to throw in the power-house switch and then returns to his own cabin, unlocks the switch box there, and throws in the switch, thus firing all the shots in the mine. After the shots have been fired, the shot firer opens the switch and locks the box.

There are always some misfires, caused by carelessness in making connections, by falls of roof deranging the wires, or by projecting portions of the roof cutting the insulation off the wires. The fire boss goes into the mine about an hour after the shots have been fired and unlocks each box, throws out the switch, and locks the box again. He then proceeds to each working place, notes what shots failed to go off, and connects them up properly. When he leaves the mine at the close of his shift he unlocks each box, throws in the switches, and

fires the shots just as a shot firer on a day shift would do. By this procedure the number of failures is reduced to a minimum. The average proportion of failures is said to be about one-half of 1 per cent, or about one failure to every 200 shots.

Figure 4 shows a plan of mine workings wired for such a method of electrical firing.

The cost of equipping a mine for firing shots by the method above outlined varies from \$1,000 to \$3,000, depending upon the size of the mine. The maintenance and operation of the system is said to cost from 1 to 3 cents per ton. It has been used successfully for some time past in some coal fields, and its adoption in other fields, where permitted by local conditions, would reduce accidents in coal mines.

SUMMARY.

The successful methods that have resulted from the efforts made in European coal-producing countries to reduce explosions in coal mines are rapidly being put in operation in the United States.

Following the example of Great Britain, Belgium, and Germany, a large number of the coal-mining States of this country are restricting the use of black blasting powder in the more dangerous coal mines. In consequence, permissible explosives are rapidly taking its place. The table on page 9 clearly shows the marked increase in the use of safer explosives during 1909.

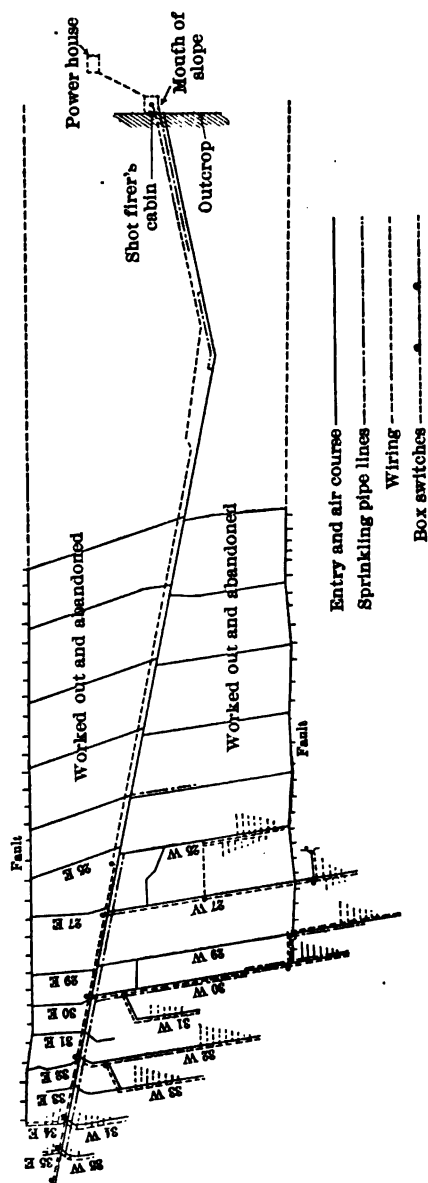


FIGURE 4.—Plan of mine workings wired for electrical firing.

In that year 8,598,027 pounds of permissible explosives, representing 31 different kinds, were used in the United States. In the mines and

quarries of Great Britain there were used, in the same year, 8,502,232 pounds of "permitted" explosive, representing 48 different kinds.

Better regulations for the storage and handling of explosives for use in coal mines are being adopted by many of the large mining companies.

The quantity of explosive to be taken into a mine is limited, in many districts, through the action of State authorities and the more progressive operators.

Special men known as shot firers are employed in many mines for loading, tamping, and firing all explosives. Some mines have adopted the excellent practice of giving these men charge of the handling of explosives.

The system of firing shots from the surface by electricity after all men are out of the mine, first adopted in Utah, is being introduced in Colorado, New Mexico, Alabama, and other States.

Coal dust or other combustible and therefore dangerous material for stemming is being replaced at mines in all parts of the country by clay or other incombustible material.

The practice of wetting the coal dust in the mines, by watering the sides and roofs of working places and by humidifying the mine air with steam jets or water sprays has spread rapidly in the past two years.

State mine inspectors enforce mining laws and the regulations governing coal mines much better now than they did a few years ago.

The measures taken in this country to prevent explosions in coal mines are becoming nearly as effective as those adopted in Europe, and the proportion of deaths caused by explosions will grow smaller as preventive measures become still more effective.

PUBLICATIONS ON MINE ACCIDENTS AND TESTS OF EXPLOSIVES.

The following Bureau of Mines publications may be obtained free by applying to the Director, Bureau of Mines, Washington, D. C.:

BULLETIN 15. Investigations of explosives used in coal mines, by Clarence Hall, W. O. Snelling, and S. P. Howell, with a chapter on the natural gas used at Pittsburgh, by G. A. Burrell, and an introduction by C. E. Munroe. 1911. 197 pp., 7 pls.

BULLETIN 17. A primer on explosives for coal miners, by Charles E. Munroe and Clarence Hall. 1911. 61 pp. 10 pls. Reprint of United States Geological Survey Bulletin 423.

BULLETIN 20. The explosibility of coal dust, by George S. Rice, with chapters by J. C. W. Frazer, Axel Larsen, Frank Haas, and Carl Scholz. 1911. 205 pp. 14 pls. Reprint of United States Geological Survey Bulletin 425.

MINERS' CIRCULAR 2. Permissible explosives tested prior to January 1, 1911, and precautions to be taken in their use, by Clarence Hall. 1911. 12 pp.

MINERS' CIRCULAR 3. Coal-dust explosions, by George S. Rice. 1911. 22 pp.

MINERS' CIRCULAR 4. The use and care of mine-rescue breathing apparatus, by J. W. Paul. 1911. 24 pp.

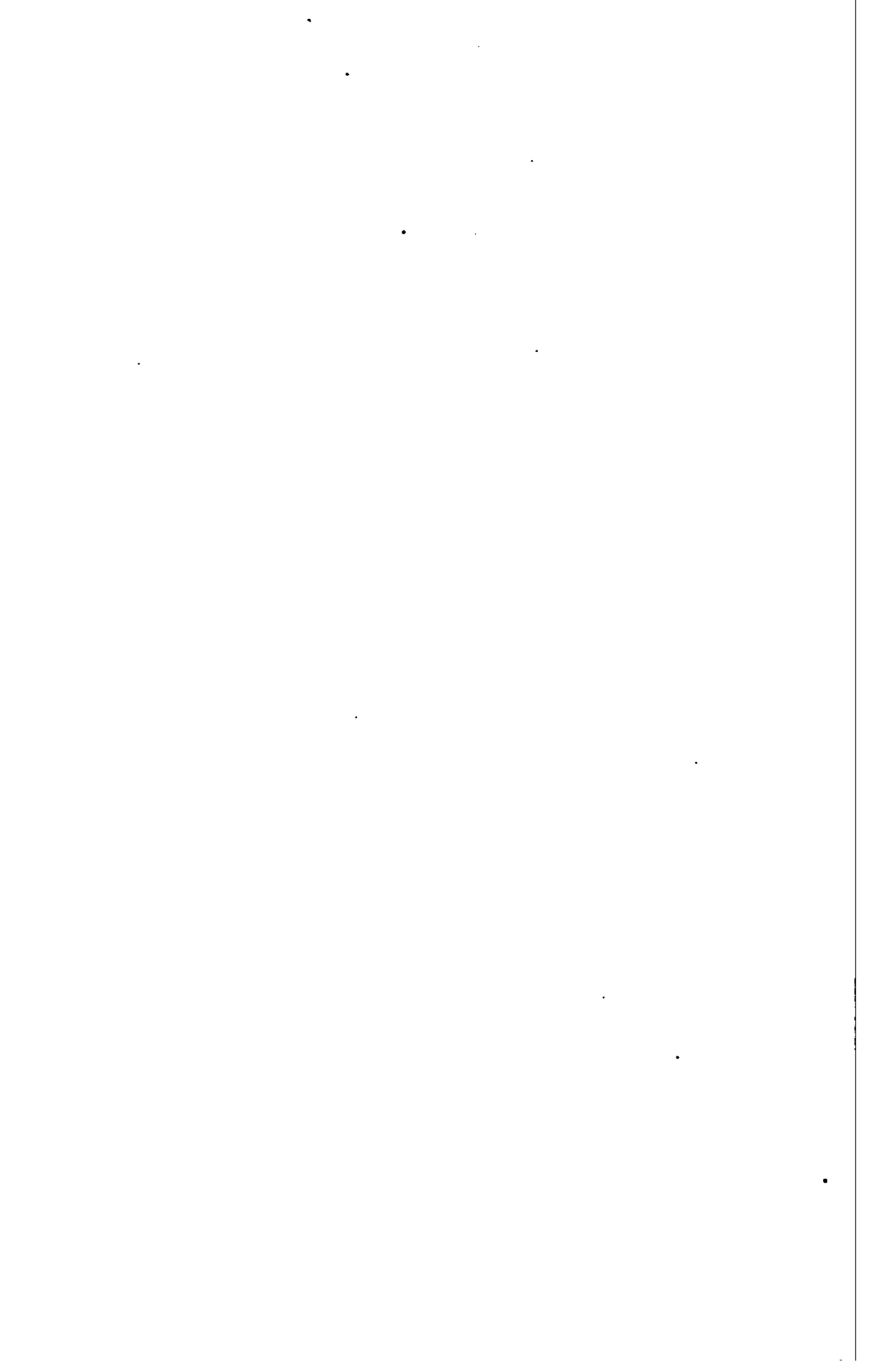
MINERS' CIRCULAR 5. Electrical accidents in mines; their causes and prevention, by H. H. Clark. 1911. 10 pp. 2 pls.

TECHNICAL PAPER 4. The electrical section of the Bureau of Mines, its purpose and equipment, by H. H. Clark. 1911. 13 pp.

TECHNICAL PAPER 6. The rate of burning of fuse, as influenced by temperature and pressure, by W. O. Snelling and W. C. Cope. 1912. 28 pp.

TECHNICAL PAPER 7. Investigations of fuse and miners' squibs, by Clarence Hall and Spencer P. Howell. 1912. 19 pp.





DEPARTMENT OF THE INTERIOR
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RIOR

APPARATUS AND METHODS

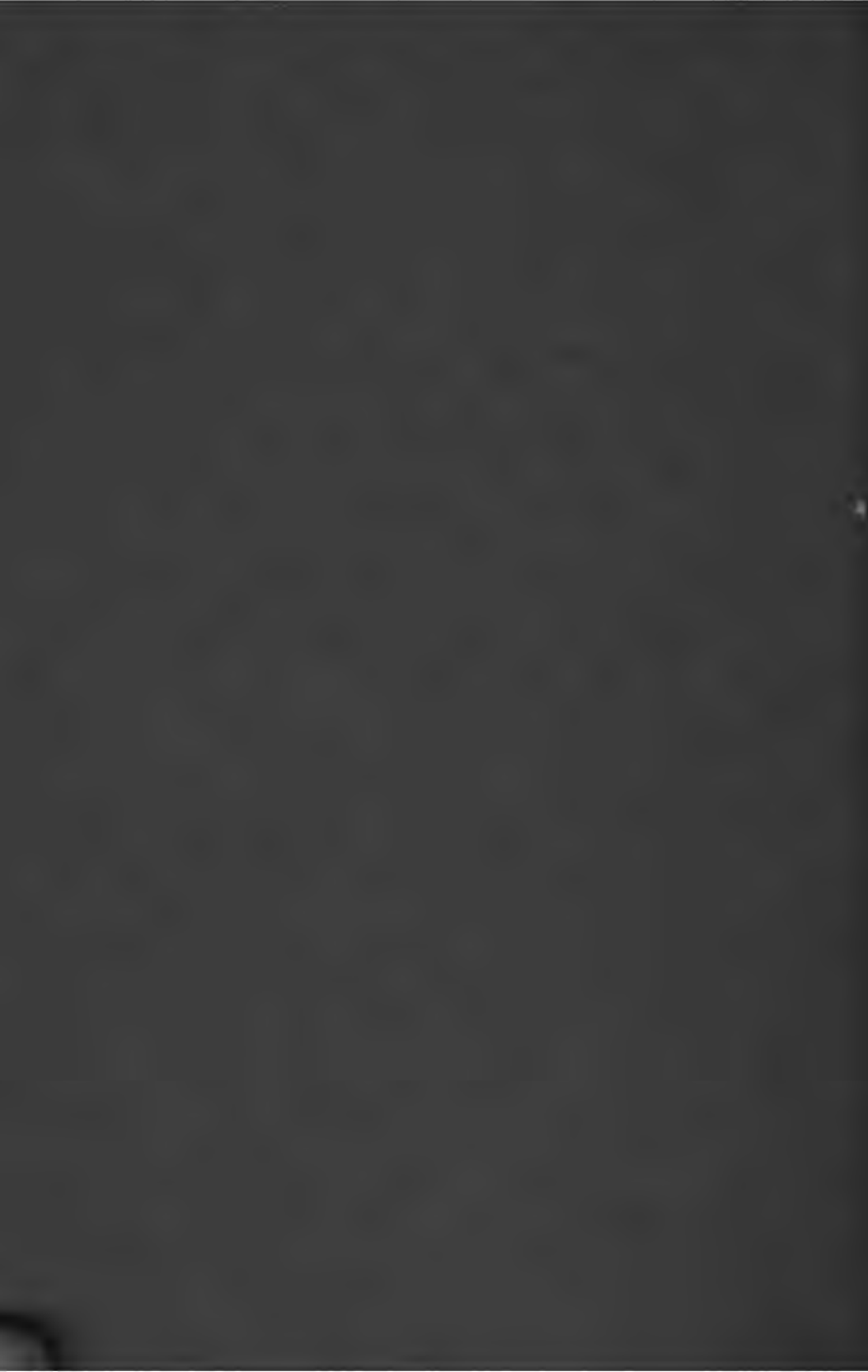
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SAMPLING AND ANALYSIS
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Bulletin 12

DEPARTMENT OF THE INTERIOR
BUREAU OF MINES
JOSEPH A. HOLMES, DIRECTOR

APPARATUS AND METHODS
FOR THE
SAMPLING AND ANALYSIS
OF FURNACE GASES

BY

J. C. W. FRAZER AND E. J. HOFFMAN



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APPARATUS AND METHODS FOR THE SAMPLING AND ANALYSIS OF FURNACE GASES.

By J. C. W. FRAZER and E. J. HOFFMAN.

INTRODUCTION.

The apparatus and methods described in the following pages were devised and used by the writers in connection with certain investigations of the technologic branch of the United States Geological Survey, especially those bearing on the improvement of furnace conditions and on efficiency in the use of fuel.

The Bureau of Mines, which has charge of such analyses and tests of fuels as were formerly made by the technologic branch of the Survey, is using the same apparatus and methods in connection with similar investigations that are being carried on at the testing plant of the bureau at Pittsburg, Pa.

The furnace conditions prevailing both in small plants and in large industrial establishments in this country are frequently far from satisfactory. If such conditions are to be improved, they must be more thoroughly understood, and means must be found to insure complete combustion of the fuel, and yet to permit operation with such an excess of air as will result in the greatest efficiency.

In this work the services of the chemist are indispensable. A very important problem is the determination of the small percentage of unburned combustible matter that escapes from the furnace in the flue gases. Under ordinary circumstances so little as 0.1 per cent of unburned combustible matter in a furnace gas is equivalent to about 1 per cent of the fuel used; and for the determination of such small percentages of gas more accurate and refined methods are required than have ordinarily been available heretofore.

The purpose of this paper to describe some apparatus and methods which have proved satisfactory to those engaged in the work of the investigations mentioned. While these methods have been used more particularly with regard to the analysis of furnace gases, the method is by no means limited to that use, and their applicability to other analyses of gases are involved.

SAMPLING OF FURNACE GASES.

The proper sampling of gases is frequently difficult when the gas mixture under investigation is not homogeneous. The problem in sampling furnace gases is further complicated by the necessity of protecting the part of the sampling apparatus introduced into the heated gases. It is not difficult to obtain a sample of gas from a given point within a furnace; but as in most cases the composition of the gas is constantly changing, some method must be provided by which the sample will represent the average composition of the gas at the point of collection during a desired period; or the sample must be collected almost instantaneously, so that it will merely represent the composition of the gas at the point and at the instant of collection.

A sample taken in either of these ways is only representative of the gas occupying a certain space surrounding the point of collection. In order to determine the average composition of the entire volume of gas it is necessary to multiply the number of samples and to distribute the points of collection in such a manner that the average of the samples will correctly represent the entire gas body. The number of samples taken should depend on the differences in composition that are presumed to exist throughout the volume of gas to be sampled. The difference in composition between samples taken at any two adjacent points of collection should not be greater than from 0.3 to 0.5 per cent.

In the following pages methods are described for the collection of "continuous" samples, taken during any convenient period of time, and of "instantaneous" samples.

CONTINUOUS SAMPLING.

The sampling of flue gases can usually be accomplished satisfactorily by using a perforated iron pipe placed in the flue at the desired point. For a sample of flue gases to be representative it should be collected from a number of points in a given cross section. As the composition of the flue gases at any instant does not vary much at different points in a given cross section, the easiest method is to use a tube with 2-mm. perforations so shaped that it will best distribute the points from which the gas is to be drawn for analysis. In case none of the holes becomes stopped, a sample drawn through such a tube by gentle suction will be sufficiently representative for most purposes.

A common method of collecting a sample is to attach a large bottle filled with water to the outside open end of such a sampling tube and then to allow the water to escape at such a rate that the gas, which replaces the water in the bottle, is collected in the desired time.

It is not advisable to sample gases that contain a considerable proportion of carbon dioxide in this way because of the great ease with which water, or even a solution of common salt, dissolves that constituent and thus tends to equalize the content of carbon dioxide in the samples collected. To illustrate: If water is saturated with furnace gas containing 12 per cent of carbon dioxide and is then used in collecting a sample from gas having a carbon-dioxide content of 8 per cent, the water saturated with gas containing 12 per cent of carbon dioxide will give up some of the dissolved gas, and the sample will be found to contain more than 8 per cent of carbon dioxide. The magnitude of the error introduced in this way will vary with conditions, but it will depend largely on the length of the sampling period and the total time of contact of the sample with the water.

The following method of sampling was devised to obviate this difficulty, as well as certain others. As it has proved satisfactory it will be described in some detail.

COLLECTION AND STORAGE OF THE SAMPLE.

The glass vessel illustrated in figure 1 is utilized both as an important part of the sampling device and as a holder for the sample after collection. The vessel should have a capacity of 150 to 250 c. c. If, when the vessel is in the position shown and is filled with mercury, the stopcocks *a* and *b* are opened the mercury will flow from the lower tube. The gas will then be drawn through the upper tube, enter the vessel at *c*, and collect above the mercury. So long as the surface of the mercury remains above *c*, the same volume of gas will be collected in each equal interval of the sampling period, so that the sample obtained will be representative. The time required for a certain amount of mercury to run from the vessel can be varied from that taken when both *a* and *b* are completely open to a period of from 8 to 10 hours, or even longer, by attaching at *d*, by means of a rubber tube, a short piece of glass tubing drawn out to a smaller diameter. By trial these short glass tubes can be made of proper diameter to deliver the mercury at almost any desired rate.

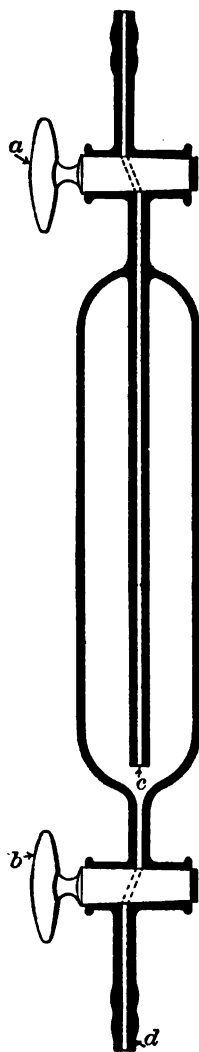


FIGURE 1.—Gas-sample collector and container.

After the sample of gas has been collected the tube above *a* is filled with mercury from a small funnel whose stem has been drawn out to a capillary. The vessel is then inverted and, by means of a rubber tube attached to a mercury reservoir, the inclosed gas is put under a pressure of about 100 mm. of mercury. When the vessel is returned to its original position, as shown in the illustration, the stopcocks *a* and *b* are mercury sealed, and there is no danger of gas leaking into or out of the vessel.

When it is desired to remove the sample for analysis the vessel is again inverted and the tube above *b* is filled with mercury and attached to the burette. By means of the mercury reservoir previously used to put the gas under pressure, the desired volume of gas is forced out of the vessel into the burette. In case the gas is to be measured over water, the tube above the stopcock *b* is filled with water instead of mercury.

In order to facilitate the safe handling of these vessels it is necessary to mount them in a portable stand, and frequently it is desirable to arrange them in batteries of two to four each. Figure 2 illustrates a convenient method of mounting four of these tubes. The stand consists of the two uprights, *A*, supporting the two shelves, *B*, and having iron rods, *C*, extending from side to side through holes bored about 3 cm. from each end of the uprights. The iron rods not only strengthen the stand, but are convenient handles. The shelves are divided longitudinally through the center and holes for the tubes are made along these lines of division. When filled with mercury the vessels should be carefully supported at both ends. Therefore the holes for the tubes are enlarged for a distance equal to about one-half the thickness of the shelf, to receive the tapering ends of the vessels, and the open spaces around these ends are filled with plaster of paris. This is done by filling the cavity in each half of the shelf independently, so that the front half of each shelf may be taken out at any time and a vessel removed without disturbing the remaining ones.

THE WATER-COOLED SAMPLING TUBE.

The portion of the sampling apparatus which is introduced into the furnace may be either a water-cooled metal tube or, better, a water-cooled quartz tube.

A type of water-cooled metal tube that has been found satisfactory is illustrated by *A*, in figure 3. The inside tube, through which the gas is collected, is kept cool by cold water, which passes through the surrounding tube and returns through the outside annular compartment. When the inner tube is of quartz, the only difference in construction is the use of asbestos packing to insure a water-tight joint at each end. An apparatus provided with a quartz tube is more fragile than

one wholly of metal, but it is preferable, as it has a greater range of utility and may even be used in the fuel bed. If the diameter of the inner tube be small enough, the end projecting from the furnace may be attached directly to the vessel receiving the sample of gas; but,

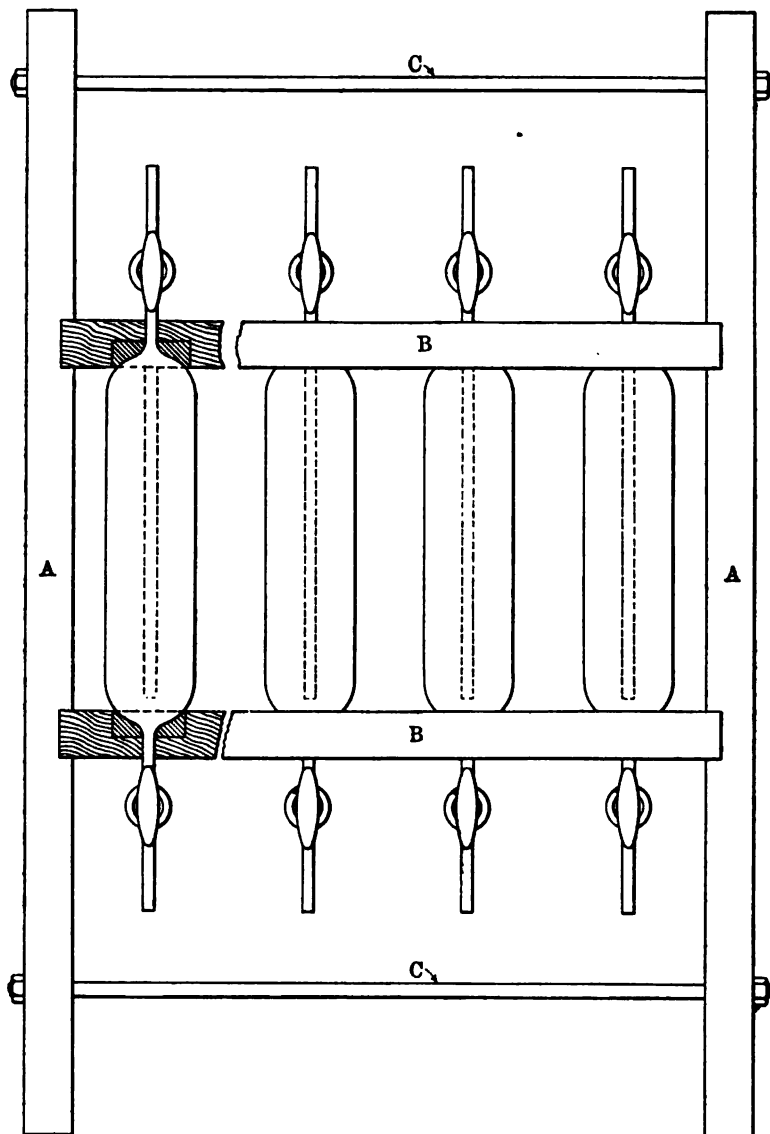


FIGURE 2.—Portable stand for gas-sample containers.

for the reasons given later, it is not generally practicable to have the diameter of the tube so small. It is preferable to connect the end of the tube with a $\frac{1}{4}$ -inch lead pipe, which is brought over to a table situated as near the furnace as possible. By connecting the other

end of the lead pipe to an aspirator, a continuous current of gas may be drawn from the furnace. If the gas is drawn through the tube rather rapidly, its composition will approximate closely that of the furnace gases at the point reached by the open end of the tube. At the table the lead pipe is perforated at a convenient point and a glass tube of small bore is inserted. This tube is connected directly with the mercury-filled sample receiver, and the sample is taken from the current of gas flowing through the lead tube.

When working in the region of the flame, it frequently happens that the water-cooled tube becomes clogged by the condensed tarry matter and by the molten ash which is carried bodily through the furnace by the rapidly moving gases. The diameter of the tube should be large enough to prevent this. Water vapor derived from moisture in the fuel and from the combustion of hydrogen and its compounds also condenses in the tube, which should be so large that the tendency to form a water column by capillarity may be easily overcome by the current of gas. Otherwise such a column may form and stop the flow of gas entirely. It is advisable to use a tube with an inside diameter of about three-sixteenths of an inch. In order to be sure that the current of gas is flowing properly through the tubes it is necessary to introduce a trap at some point beyond that at which the sample is taken from the lead pipe. The trap is simply a wash bottle containing water through which the gas bubbles on its way to the aspirator, the rate of bubbling roughly indicating conditions in the tubes. Figure 3 illustrates the entire sampling system.

As previously stated, a sample taken in this way will represent only the average conditions prevailing during the period of collection in a certain space about the point reached by the end of the sampling tube. In order to ascertain conditions throughout the furnace, or throughout any given portion of it, it is necessary to multiply and properly distribute the number of points from which samples are taken. As many sampling tubes as desired may be inserted in the furnace at different points, and separate samples collected simultaneously. The same aspirator may be used to obtain all the samples, the lead pipes attached to the different sampling tubes being led to the table on which are placed the vessels to receive the samples.

INSTANTANEOUS SAMPLING.

Frequently it is desirable to ascertain the composition of gases at some point in a furnace at a certain definite instant. Such would be the case when there might be reason to suspect that the gases would decompose during a period of continuous sampling or when it might be desired to study the progress of reactions in the furnace. This method is employed by certain investigators in collecting samples

from the flame of burning gases and from the explosion flame of coal dust, etc. Further, the use of this method enables one to obtain certain information concerning furnace conditions which he might not be able to obtain by the use of the continuous-sampling method. It was for the purpose of collecting instantaneous samples that the device shown in figure 4 was constructed.

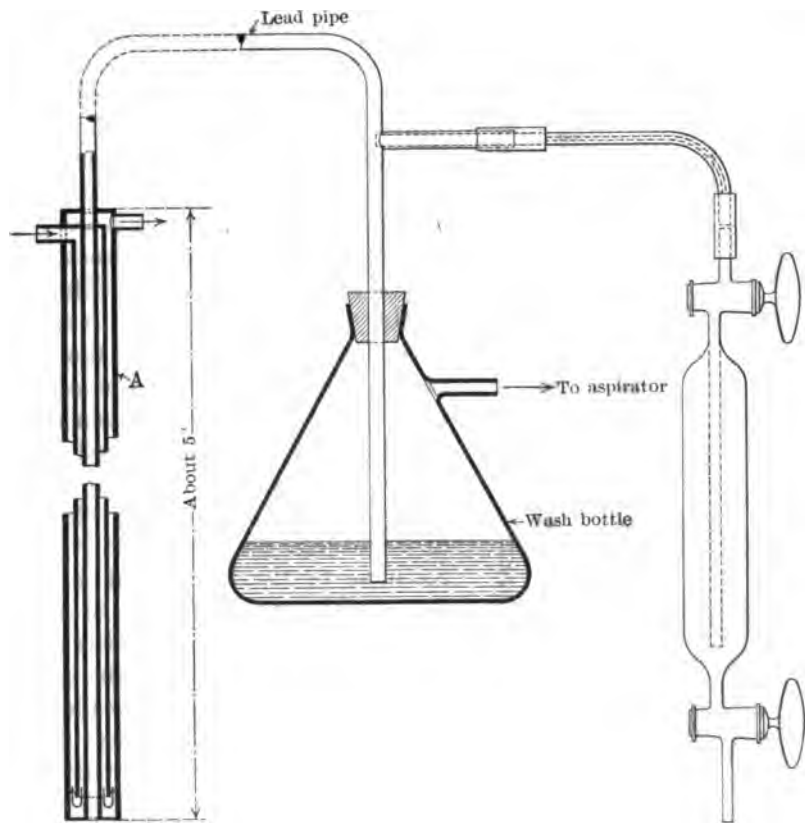


FIGURE 3.—Arrangement of apparatus for sampling furnace gases.

DESCRIPTION OF APPARATUS.

The apparatus consists of a quartz sampling tube, A, of 100-c. c. capacity, immersed in water contained in the steel tube B, which is 1.2 meters in length and 10 cm. in diameter. At each end the vessel terminates in a thick-walled quartz tube, 1-mm. bore, provided with a stopcock, as shown. One of the tubes, *a*, extends 150 mm. beyond the stopcock, and the open end projects beyond the end of B. An enlargement, *b*, 60 mm. from the stopcock, gives a firmer hold for the cement of litharge and glycerin with which the cavity in the collar *c* is filled; in this way *c* is fastened permanently to *a*. The device

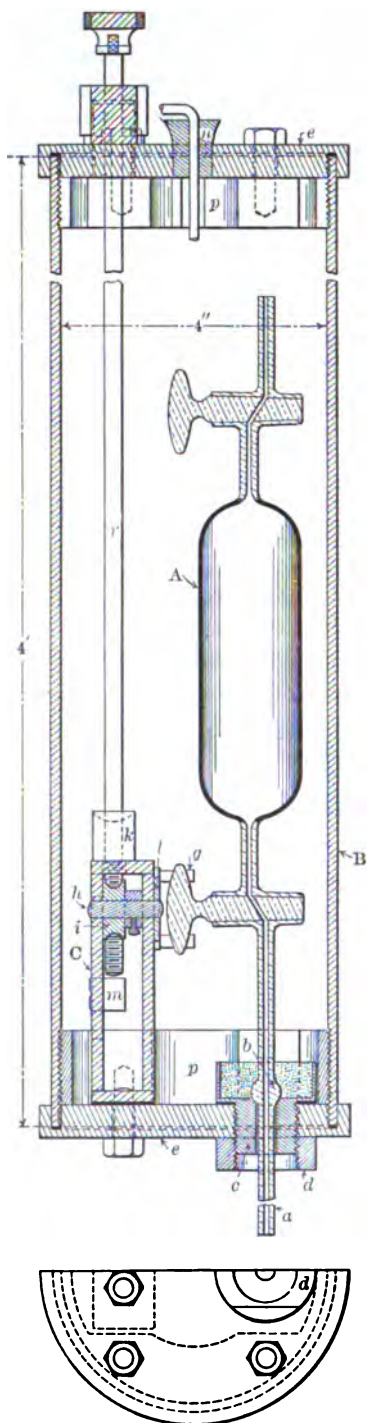


FIGURE 4.—Apparatus for taking instantaneous gas samples.

for opening and closing the sampling tube from the outside is made of brass and is supported by the two end pieces *e* of the steel tube B. It consists of the brass frame C, in which is supported the mechanism for turning the stopcock. This consists of the brass shaft *h*, on which is set the wheel *i* and, beneath the frame, the brass plate *l*, carrying four projections, *g*, which fit around the handle of the stopcock, as shown. In order to avoid straining the stopcock in turning, which might occur if *h* were not centered above the stopcock, the pieces *g* are small rollers. The face of the wheel *i* is threaded to engage with the threaded end of the brass rod *r*. The piece *k* serves as a guide for the brass rod and affords a means of adjusting the threaded end of the rod to the face of *i*. The adjustment is accomplished by having the hole in *k* through which the rod *r* passes eccentric to the bearing of *k* in C. A movable stop, *m*, can be set to limit the rotation of *i* and the extent to which the stopcock may be turned. The steel tube B is threaded internally at each end to receive the threaded steel pieces *p*, which are tapped to receive bolts which secure the end pieces *e*. The arrangement of this portion of the apparatus can be seen best by referring to the semi-circular drawing which shows the plan of the end introduced into the furnace. All joints leading to the interior of B are made tight by lead washers, on which the two surfaces are tightly drawn by means of bolts.

OPERATION OF APPARATUS.

Before collecting a sample, the end piece *e*, which is to carry the vessel A, is removed by unscrewing the

bolts which bind it to B. The quartz tube is placed in position and the nut tightened. During this operation *c* is prevented from rotating by two small dowel pins which enter holes provided for them in *e*. By trial *m* is adjusted so that when the rod *r* reaches it the handle of the stopcock is rotated 90°. It is customary to adjust the stopcock so that communication with A is established when *r* is against *m* as shown. The withdrawal of *r* then closes the stopcock. Having the stopcock properly adjusted, the end piece *e* is bolted to B, and the latter is suspended in a vertical position by a handle clamped on the other end, and not shown in the illustration. While in a vertical position the rod *r* is introduced through the brass bushing at the upper end and finds its way without difficulty into the hole in *k*. The steel tube B is filled with water through *n*, which is then closed by a perforated rubber stopper, through which passes a short glass tube bent at a right angle and having its projecting end directed upward when the apparatus is in use. This tube relieves the internal pressure when the temperature of the water rises.

The apparatus is then taken to a vacuum pump and placed in a horizontal position in two semicircular rests which prevent its moving while it is connected with the pump. When the tube A is exhausted, and while it is still connected with the pump, *r* is withdrawn so far that the threaded end leaves *i* but remains in *k*. This closes the stopcock. The time at which *r* and *i* become disengaged can be readily determined by the increased ease with which *r* moves. The pump is disconnected, and as soon as convenient the apparatus is introduced into the furnace, the open end of the tube *a* being placed at the point from which the sample is desired. The stopcock is opened by pushing in the rod *r* until it is in contact with *m*, and after the short time required for A to fill the rod is withdrawn and the stopcock thus closed.

As the sampler remains in the furnace only about 30 seconds, and as the quantity of water in B is considerable, it is unnecessary to provide for a circulation of this water. In collecting a sample it was found that the temperature of water in B did not rise more than 10° even when the tube was inserted in the hottest part of the furnace.

After removing the apparatus from the furnace, the water is drained from B, the rod *r* is withdrawn, the nut *d* is started to insure its subsequent easy removal, and the end piece *e*, carrying *a*, is removed. Then *d* is taken entirely off, and after the two glass tubes have been filled up to the stopcocks with mercury the sample in A is transferred to one of the holders illustrated in figure 1. Since it requires about 30 minutes for the complete operation of collecting a sample in this way, a series of samples is generally collected and stored in the holders before making the analyses.

The vacuum pump employed for evacuating the quartz sampling tubes is a Duplex A Geryk pump with which it is possible to secure a pressure as low as a few tenths of a millimeter of mercury in a short time. The stopcocks of the quartz tubes are always carefully polished with the finest emery and tested before using. It has been found that when one of these tubes is evacuated as completely as possible and then allowed to stand closed and detached from the pump for about 30 minutes, no leakage of air into the bulb can be detected by opening the stopcock with the end of the tube immersed in mercury.

DETERMINATION OF MOISTURE AND NITRIC OXIDE

The results of the analyses of samples collected instantaneously show the conditions prevailing temporarily at the points of collection. If it is desired to coordinate the results of analyses with conditions of furnace manipulation, it is necessary to resort to the continuous-sampling method, which gives the average composition of the gases during the period of sampling. However, the instantaneous method provides means of obtaining data on two points, which data can be obtained with difficulty, if at all, by the continuous method.

With the apparatus illustrated in figure 4, the amount of water vapor accompanying a gas sample can be easily determined by absorbing the moisture and weighing it. Likewise, the presence of traces of nitric oxide in furnace gases has been shown by using a simple modification of the method just described. The determination of nitric oxide is accomplished as follows:

The sampling tube used is a water-cooled quartz tube similar to the one illustrated in figure 3. An evacuated 8-liter bottle is used as a receiver for the gas sample, in place of the quartz vessel A shown in figure 4. Two glass tubes, each provided with a carefully ground stopcock, pass through two holes in a rubber stopper which closes the bottle. One of these tubes extends almost to the bottom of the bottle; the other, which ends just below the stopper, is bent at a right angle and connected directly with the water-cooled quartz tube. This arrangement permits the easy removal from the bottle of the whole or part of the gas sample, or the introduction of any desired liquid into the bottle without exposing the contained gas. The stopper is completely covered with sealing wax to aid in making the bottle gas-tight. After the introduction of the sampling tube into the furnace, the evacuated bottle is filled by simply opening the proper stopcock.

After the collection of the sample, an excess of an alkaline solution of potassium permanganate is introduced into the bottle and allowed to stand for 24 hours. At the end of that time the solution is withdrawn and the free ammonia distilled off. Potassium hydroxide and

fine aluminum powder are then added and the mixture allowed to stand several hours, after which the ammonia formed is distilled into standard sulphuric acid.

The presence of nitric oxide can be demonstrated qualitatively by introducing a solution of starch and potassium iodide into the bottle directly after the collection of the sample. The blue color does not appear immediately, probably owing to the presence of sulphur dioxide in the gas, but in a short time the color becomes noticeable and is soon quite pronounced.

In the gases examined the quantity of the nitric oxide found varied from 0.015 to 0.031 per cent. At the time of taking these samples the furnace conditions were not favorable to the formation of this constituent, and there is reason to believe that the percentage of nitric oxide in the furnace gases is frequently much greater. It would require further work to determine whether enough nitric oxide to be of importance is ever present in furnace gases, and also to determine whether the source of this nitric oxide is the nitrogen of the fuel or the nitrogen of the air. The presence of relatively small quantities of the constituent is of some importance, since the formation of the gas is accompanied by an absorption of heat.

ANALYSIS OF THE SAMPLE.

The sample having been collected, its analysis is performed most conveniently by the method of Hempel. The use of mercury in the burette is to be preferred to that of water; but whichever is used, the burette should always be provided with a water jacket to avoid errors due to sudden changes in temperature. While the ordinary Hempel burette is sufficiently accurate for most purposes, it does not enable the observer either to detect or to determine changes in volume amounting to 0.1 c. c. or less, since the error in reading the burette itself can not be less than this volume.

As already stated, it is a matter of some importance to be able to measure smaller percentages of combustible gases than can be determined by the ordinary Hempel method, since the flue gases are so diluted that a small percentage of combustible matter in them corresponds to a much greater percentage of the fuel. In order to determine these small percentages of combustible gases, a very accurate method is required. The apparatus described by Hempel for exact gas analysis, which provides a tube for compensating errors due to variations in the pressure and temperature of the atmosphere (the principle of Pettersson*), is in certain respects unsatisfactory. In this instrument a considerable portion of the air in the compensator and an approximately equal volume of the gas being meas-

* *Zeitschr. für anal. Chemie*, vol. 25, pp. 467-484.

ured are not inclosed in the water jacket, and while there is a tendency to equalize any error due to this arrangement, the compensation is perfect only when the total volumes of the two gases are equal. Further, the burette itself can not be read with sufficient accuracy.

In 1900 A. H. White^a published a description of an apparatus devised to obviate these difficulties. In this apparatus the principle of automatic compensation for changes in temperature and pressure as suggested by Pettersson and modified by Hempel and others is utilized in an improved form. The measuring portion of this apparatus consists of two limbs, suggested by the burette of Otto Bleier,^b one a series of bulbs to contain the larger portion of the measured gas, the other a long straight tube of small capacity on whose scale all final readings of volume are made.

It was in order to obviate the same difficulties and to be able to measure all possible changes in volume that the apparatus described below was designed by the writers. Since this apparatus was devised entirely independently and used for some time before their attention was called to the work of Bleier and of White, and since it is believed to possess certain advantages over other similar forms of apparatus, its description is given here, but without making any claim of priority as to its essential principles.

DESCRIPTION OF APPARATUS.

The apparatus, illustrated in figure 5, consists of the burette A and the automatic compensating device B. The measuring portion of the burette A and the whole of the compensator B are inclosed in the water jacket C. The measuring portion of the burette consists of the two limbs *a* and *b*, the graduated portions of which are 66 cm. long and united at the top in an inverted Y-shaped connection to which a Greiner-Friedrich two-way stopcock is attached. Through this stopcock communication can be made with either of two short, thick-walled tubes of small bore, one of which is connected with the compensating device. Outside the water jacket C, at the lower end, is a second Y tube, each of whose limbs is provided with a stopcock and attached by rubber connections to the projecting ends of the limbs *a* and *b* as shown. To the lower end of the Y tube is attached heavy rubber tubing connected with a mercury reservoir not shown in the illustration.

The tube *a* consists of a series of 10 bulbs, each having a capacity of 10 c. c. between the two graduation marks immediately above and below it. The straight glass tube *b* has an internal diameter of about 4.5 mm., and its graduated part has a total capacity of 10.1 c. c. The

^a Jour. Am. Chem. Soc., vol. 22, 1900, p. 343.

^b Ber. Deutsch. chem. Gesell., vol. 30, 1897, pp. 2759, 3128.

beginning of the graduated portion of each limb of the burette is at *c*. The compensator B, while utilizing the principle of Pettersson of counterbalancing the pressure of the gas to be measured with that of a constant mass of air occupying a constant volume, is arranged in a somewhat different form from Pettersson's device. The confined air, whose pressure at a constant volume is equalized by that of the gas to be measured, is contained in the bulb *d* and above the mercury surface *e* in the tube *g*, which forms the lower termination of the bulb.

The glass tube *f* is connected at its upper end with one of the communications through the stopcock of the burette and is sealed into the top of the bulb *d*. Its other end extends nearly to the bottom of *g* and opens beneath the surface of the mercury. It is centered in *g* at two points—*h*, which is 25 mm. above the surface of the mercury, and *k*, about 50 mm. from its lower end—by three small glass projections sealed on the outside of it at each of these points. The diameters of the tubes *f* and *g* are proportioned so that the distance from the inside of *g* and the outside of *f* is as nearly as possible equal to the internal diameter of *f*. The distance from the surface of the mercury in *f* to the stopcock of the burette should be as short as possible, since the gas in this portion of the tube must be drawn into the burette by decreasing the pressure. The depth of mercury in the compensator should be sufficient to require about one-third of

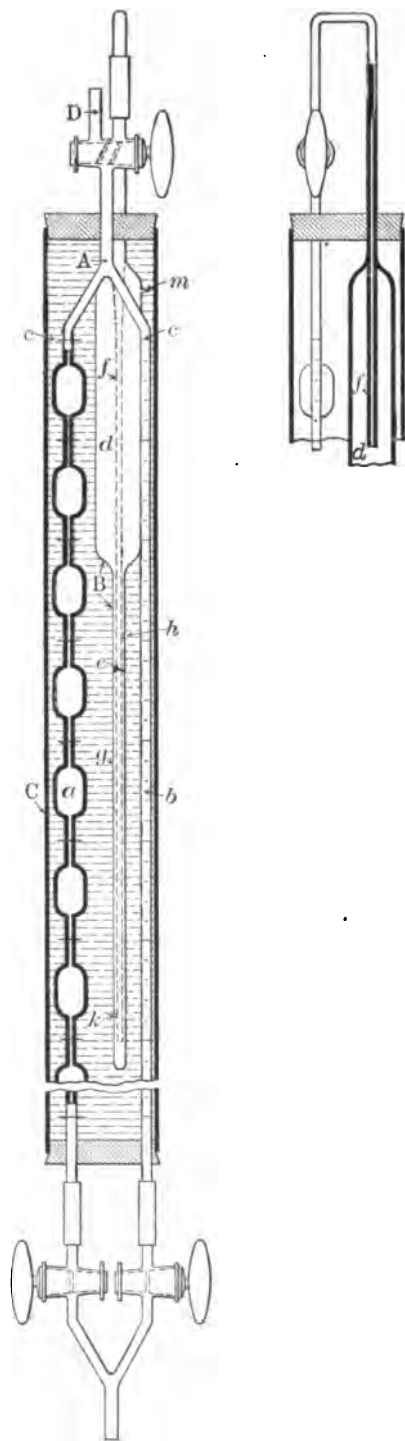


FIGURE 5.—Apparatus for exact gas analysis.

an atmosphere excess pressure in the burette to force gas from the latter into the compensator, but the quantity of mercury should not be so great that its expansion or contraction will cause an appreciable change in the volume of the inclosed air. The tube *f* should extend nearly to the bottom of the compensator in order to displace as much mercury as possible.

The compensator may be filled so that the readings on the burette are the correct volumes of the gas at 0° C. and 760 mm. pressure, or it may be closed under known conditions of temperature and pressure and the readings corrected to standard conditions. The latter method is sufficient for most purposes, and when it is used the compensator may be closed by replacing the seal at *m* by a small tube and stopcock.

OPERATION OF APPARATUS.

To obtain a reading of the volume of a gas in the burette, it is first necessary to stir the water in the jacket C and allow a few minutes to elapse in order to be sure that the gas is at the temperature of the surrounding water.* Then by drawing nearly all the gas into the limb *a* and adjusting the pressure to approximately that of the atmosphere, the number of bulbs the gas will fill completely when at the pressure of the air in the compensator may be ascertained. Having determined this, the mercury in the burette is brought exactly to the level of the graduation beneath the last bulb completely filled in the trial experiment and the stopcock at the bottom of the limb is closed. The remaining fraction of a bulb full of gas is then made to enter the limb *b*, and the two-way cock is turned so as to place the burette in communication with the compensator. The pressure of the gas in the burette is then adjusted, by means of the mercury reservoir, until the two surfaces of mercury in the compensator are on the same level. The pressure of the gas in the burette is then equal to that of the gas in the compensator. The stopcock at the bottom of *b* is then closed and the reading of the burette taken. To this reading is added the predetermined capacity, *K*, of that part of the apparatus between the graduated portion of each limb of the burette and the mercury meniscus in the tube *f*. As each constituent of the gas is determined by the difference in the burette readings, before and after an absorption, this constant capacity, *K*, does not enter into the determination of the amount absorbed. But it is necessary to apply this correction to obtain the initial volume of the gas unless exactly sufficient nitrogen to fill this part of the apparatus is taken into the burette previous to the introduction of the sample.

* The water is stirred by blowing air through a glass tube, not shown in the illustration, which extends nearly to the bottom of the water jacket.

The reading of the burette requires some practice to secure the best results, but with experience it can be accomplished quite readily and with extreme accuracy. The gases are measured in the moist condition, and the quantity of water introduced into the burette and compensator to effect this must be only sufficient to moisten the walls of the glass tubes. If there is sufficient water in the compensator to drain down upon the mercury, the accurate adjustment of the mercury surfaces is rendered difficult; while too much water in the burette is likely to stop up the narrow tubes connecting the bulbs, and in this way to interfere seriously with the distribution of pressure on the gas in the burette and consequently with the equalization of the pressures in the burette and compensator.

With this apparatus a complete analysis of gas may be made in the usual way by connecting the burette in turn with various absorption pipettes. It is, however, more convenient and accurate to connect the burette permanently with the desired pipettes. By this arrangement errors due to the repeated making and breaking of connections are avoided. The entire system may be conveniently mounted on the same wooden support, and for the analyses made thus far such an arrangement has been employed.

A series of three pipettes is connected with the tube D by means of fine capillary tubing in the manner generally employed with the Orsat apparatus. These pipettes consist of a combustion pipette provided with an electrically heated platinum coil of the type recommended by L. M. Dennis, an absorption pipette containing a solution of potassium hydroxide for the absorption of carbon dioxide, and a pipette containing phosphorus for the determination of oxygen. In order to avoid error due to the capacity of the connecting tubes, these are made as short as possible, and their internal diameter is not greater than 0.4 mm. All error from this source can be eliminated entirely either by actually determining the capacity of the connections or by filling them and that part of the apparatus between the ungraduated portions of the burette and the mercury meniscus in the tube *f* with pure nitrogen before introducing the sample into the burette.

DEGREE OF ACCURACY ATTAINABLE.

The limb *b* of the burette on which the readings are made is graduated in hundredths of 1 c. c., and tenths of these divisions can be estimated quite accurately. Experiments have shown that 50 to 100 c. c. of gas can be measured accurately to 0.01 c. c. With volumes less than 50 c. c. the readings can be more precise, as a more accurate adjustment of the mercury surfaces in the compensator is possible. The following examples show the degree of accuracy which may be safely claimed for the apparatus.

(1) A volume of nitrogen passed in succession into the pipette containing potassium hydroxide and the pipette containing phosphorus was drawn into the burette and its volume determined. The two readings were 66.516 and 66.514 c. c. The gas was then passed into the pipette containing potassium hydroxide, where it was allowed to stand two minutes, after which it was returned to the burette and measured. The duplicate readings were 66.516 and 66.506 c. c. After standing in the pipette containing phosphorus for three minutes, three readings showed 66.513, 66.526, 66.521 c. c. The gas was then passed back and forth several times from the burette to the combustion pipette, after which the reading was 66.526 c. c.

(2) A sample of air was collected and the carbon dioxide and oxygen were determined. In this case the capillary connections and that part of the apparatus between the graduated portion of each limb of the burette and the mercury meniscus in the tube *f* were filled with nitrogen before introducing the sample. In analysis (*a*) the volume of air used was 60.506 c. c. and in analysis (*b*) it was 60.906 c. c. The analyses gave: (*a*), $\text{CO}_2=0.057$ per cent, $\text{O}_2=20.866$ per cent; (*b*), $\text{CO}_2=0.046$ per cent, $\text{O}_2=20.834$ per cent.

The data obtained in this way from the analysis of air may be used for determining K^e and the capacity of the capillary connections. To do this the whole system is filled with air whose analysis is accurately known, and the volume included in the graduated portions of the burette is read. The carbon dioxide and oxygen are then absorbed, and the contractions observed complete the data required for determining the total volume of air introduced into the apparatus, and consequently the volume contained in the capillary connections and that part of the apparatus between the graduated portions of the burette and the mercury meniscus in the tube *f*.

The above description of the manipulation of the apparatus involves the use of a leveling bottle attached to the lower Y tube. With care the pressures in the burette and compensator can be equalized accurately by the use of the leveling bottle alone, but the equalizing can be accomplished more rapidly and with greater precision by making the final adjustments by means of a screw plunger fitting tightly in its bearings in a closed steel reservoir filled with mercury. The steel reservoir has two openings, one connected with the Y tube by means of a short piece of rubber tubing, and the other connected with the leveling bottle through a longer rubber tube. A glass stopcock is placed between the reservoir and the leveling bottle, by means of which communication between the two is cut off previous to the final adjustment of pressure by means of the screw plunger. The steel reservoir also serves as a trap for much of the dirt which becomes loosened from the inner walls of the connecting rubber tubes, and would otherwise find its way into the burette.

CALIBRATION APPARATUS.

The principle of the screw-plunger reservoir has been utilized in constructing a device for calibrating burettes and pipettes, or other similar apparatus used in volumetric work. This device is illustrated in figure 6. It consists of the steel plunger A, to which is attached the steel cylinder B, whose circumference at *a* is divided into 100 divisions, the steel piece D, in which is cemented the glass tube E, and the steel piece C, which is threaded to receive the lower end of A.

The plunger A is the part that has to be made with the greatest care. Its diameter is so related to the pitch of the thread cut on its lower end that one complete revolution changes the capacity of E by 1 c. c. The plunger is finished by careful grinding.

The joint around A is packed with a hard-rubber washer, F, turned to fit between the two pieces C and D and the face of A. The shape of the shoulders of C and D at this point is such as will tend to produce a tight joint with A when C and D are screwed together.

The glass tube E is cemented in D with a mixture of litharge and glycerin. The exposed surfaces of the cement between E and D are subsequently coated several times with a solution of rubber in carbon bisulphide. After each application the rubber coating is hardened until a tight joint is obtained which will withstand the pressure of the liquid within.

The motion of A changes the capacity of E by the volume of that portion of the plunger entering or leaving the hard-rubber ring F. The scale on which this motion is read is a series of horizontal lines drawn on the face of C and indicating the number of whole revolutions of the plunger. The divisions of the circumference of B at *a* show the fractions of a revolution in hundredths. A second scale is superimposed on the first scale on C and consists of a series of 11 vertical lines spanning the same distance as 10

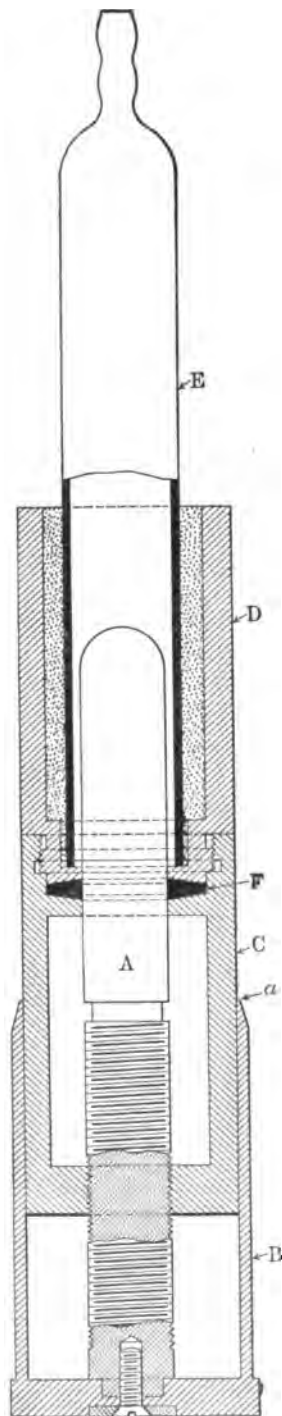


FIGURE 6.—Calibration apparatus.

lines on B. This vernier subdivides the divisions on B and permits the reading of the rotation of A to one one-thousandth of a revolution. When A is ground so that it displaces 1 c. c. when advanced one turn, the readings on the vernier are thousands of a cubic centimeter. However, the grinding can rarely be done with sufficient accuracy for each revolution of the plunger to represent exactly 1 c. c., but to calibrate the apparatus it is only necessary to fill E with mercury and weigh the quantity displaced at each complete revolution. The instrument should be calibrated at various temperatures, and to insure accurate readings it is necessary either that the lower portion of B have a grip of nonconducting material or that the operator wear a woolen glove on the hand used for manipulating the instrument.

By means of this apparatus instruments may be calibrated with either mercury or water. If water is used the water is introduced on top of enough mercury to prevent the water from coming in contact with the plunger at any time.

The following table gives the data obtained by calibrating an ordinary 50 c. c. burette with water by means of the above described instrument. A previous determination of the volume of mercury displaced by each revolution of the plunger showed that it was so nearly 1 c. c. that it could be assumed as such in a calibration of this kind. Starting with zero on the burette scale, each successive reading of the burette was made after the plunger had been withdrawn exactly two revolutions. As shown in the table the two series of readings made on the same burette agree closely. The time required to obtain both series of results was less than one hour.

Calibration data.

Series 1.		Series 2.	
Readings on the burette.	Differences.	Readings on the burette.	Differences.
0.00	0.00
2.01	2.01	2.01	2.01
4.01	2.00	4.01	2.00
5.99	1.99	5.99	1.98
7.97	1.98	7.98	1.99
9.94	1.97	9.94	1.96
11.93	1.99	11.94	2.00
13.92	1.99	13.93	1.99
15.91	1.99	15.92	1.99
17.90	1.99	17.91	1.99
19.89	1.99	19.90	1.99
21.88	1.99	21.88	1.98
23.85	1.97	23.85	1.97
25.82	1.97	25.83	1.98
27.81	1.99	27.82	1.99
29.81	2.00	29.82	2.00
31.80	1.99	31.80	1.98
33.78	1.98	33.79	1.99
35.74	1.96	35.77	1.98
37.73	1.99	37.74	1.97
39.72	1.99	39.72	1.98
41.69	1.97	41.69	1.97
43.68	1.99	43.68	1.99
45.65	1.97	45.65	1.97
47.63	1.98	47.65	2.00
49.65	2.02	49.66	2.01

PUBLICATIONS ON FUEL TESTING.

The following publications, except those to which a price is affixed, can be obtained free by applying to the Director of the Bureau of Mines, Washington, D. C. The priced publications can be purchased from the Superintendent of Documents, Government Printing Office, Washington, D. C.

PUBLICATIONS OF THE BUREAU OF MINES.

BULLETIN 1. The volatile matter of coal, by H. C. Porter and F. K. Ovitiz. 1910. 56 pp., 1 pl.

BULLETIN 2. North Dakota lignite as a fuel for power-plant boilers, by D. T. Randall and Henry Kreislinger. 1910. 42 pp., 1 pl.

BULLETIN 3. The coke industry of the United States as related to the foundry, by Richard Moldenke. 1910. 32 pp.

BULLETIN 4. Features of producer-gas power-plant development in Europe, by R. H. Fernald. 1910. 27 pp., 4 pls.

BULLETIN 5. Coking and washing tests of coal at Denver, Colo., July 1, 1908, to June 30, 1909, by A. W. Belden, J. W. Groves, K. M. Way, and G. R. Delamater. 1910. 62 pp.

BULLETIN 6. Coals available for illuminating-gas manufacture, by A. H. White and Perry Barker. 1911. (In press.)

BULLETIN 7. Essential factors in the formation of producer gas, by J. K. Clement, L. H. Adams, and C. N. Haskins. 1911. 58 pp., 1 pl.

BULLETIN 8. The flow of heat through furnace walls, by W. T. Ray and Henry Kreislinger. 1911. 32 pp.

BULLETIN 9. Recent development of the producer-gas power plant in the United States, by R. H. Fernald. 1910. 82 pp. Reprint of United States Geological Survey Bulletin 416.

BULLETIN 11. The purchase of coal by the Government under specifications, by G. S. Pope. 1910. 80 pp. Reprint of United States Geological Survey Bulletin 428.

PUBLICATIONS OF THE UNITED STATES GEOLOGICAL SURVEY.

BULLETIN 261. Preliminary report on the operations of the coal-testing plant of the United States Geological Survey at the Louisiana Purchase Exposition, in St. Louis, Mo., 1904; E. W. Parker, J. A. Holmes, M. R. Campbell, committee in charge. 1905. 172 pp. 10 cents.

PROFESSIONAL PAPER 48. Report on the operations of the coal-testing plant of the United States Geological Survey at the Louisiana Purchase Exposition, St. Louis, Mo., 1904; E. W. Parker, J. A. Holmes, M. R. Campbell, committee in charge. 1906. In three parts. 1492 pp., 13 pls. \$1.50.

BULLETIN 290. Preliminary report on the operations of the fuel-testing plant of the United States Geological Survey at St. Louis, Mo., 1905, by J. A. Holmes. 1906. 240 pp. 20 cents.

BULLETIN 323. Experimental work conducted in the chemical laboratory of the United States fuel-testing plant at St. Louis, Mo., January 1, 1905, to July 31, 1906, by N. W. Lord. 1907. 49 pp. 10 cents.

BULLETIN 325. A study of four hundred steaming tests made at the fuel-testing plant, St. Louis, Mo., 1904, 1905, and 1906, by L. P. Breckenridge. 1907. 186 pp. 20 cents.

BULLETIN 332. Report of the United States fuel-testing plant at St. Louis, Mo., January 1, 1906, to June 30, 1907; J. A. Holmes, in charge. 1908. 299 pp. 25 cents.

BULLETIN 334. The burning of coal without smoke in boiler plants; a preliminary report, by D. T. Randall. 1908. 26 pp. 5 cents. (See Bull. 373.)

BULLETIN 336. Washing and coking tests of coal and cupola tests of coke, by Richard Moldenke, A. W. Belden, and G. R. Delamater. 1908. 76 pp. 10 cents.

BULLETIN 339. The purchase of coal under Government and commercial specifications on the basis of its heating value, with analyses of coal delivered under Government contracts, by D. T. Randall. 1908. 27 pp. 5 cents. (See Bull. 428.)

BULLETIN 343. Binders for coal briquets, by J. E. Mills. 1908. 56 pp.

BULLETIN 362. Mine sampling and chemical analyses of coals tested at the United States fuel-testing plant, Norfolk, Va., in 1907, by J. S. Burrows. 1908. 23 pp. 5 cents.

BULLETIN 363. Comparative tests of run-of-mine and briquetted coal on locomotives, including torpedo-boat tests and some foreign specifications for briquetted fuel, by W. F. M. Goss. 1908. 57 pp., 4 pls.

BULLETIN 366. Tests of coal and briquets as fuel for house-heating boilers, by D. T. Randall. 1908. 44 pp., 3 pls.

BULLETIN 367. Significance of drafts in steam-boiler practice, by W. T. Ray and Henry Kreisinger. 1909. 61 pp.

BULLETIN 368. Washing and coking tests of coal at Denver, Colo., by A. W. Belden, G. R. Delamater, and J. W. Groves. 1909. 54 pp., 2 pls. 10 cents.

BULLETIN 373. The smokeless combustion of coal in boiler plants, by D. T. Randall and H. W. Weeks. 1909. 188 pp. 20 cents.

BULLETIN 378. The purchase of coal under Government specifications, by J. S. Burrows. 1909. 44 pp. 10 cents. (See Bull. 428.)

BULLETIN 382. The effect of oxygen in coal, by David White. 1909. 78 pp., 3 pls. 25 cents.

BULLETIN 385. Briquetting tests at the United States fuel-testing plant, Norfolk, Va., 1907-8, by C. L. Wright. 1909. 41 pp., 9 pls.

BULLETIN 392. Commercial deductions from comparisons of gasoline and alcohol tests on internal-combustion engines, by R. M. Strong. 1909. 38 pp.

BULLETIN 393. Incidental problems in gas-producer tests, by R. H. Fernald, C. D. Smith, J. K. Clement, and H. A. Grine. 1909. 29 pp. 5 cents.

BULLETIN 402. The utilization of fuel in locomotive practice, by W. F. M. Goss. 1909. 28 pp.

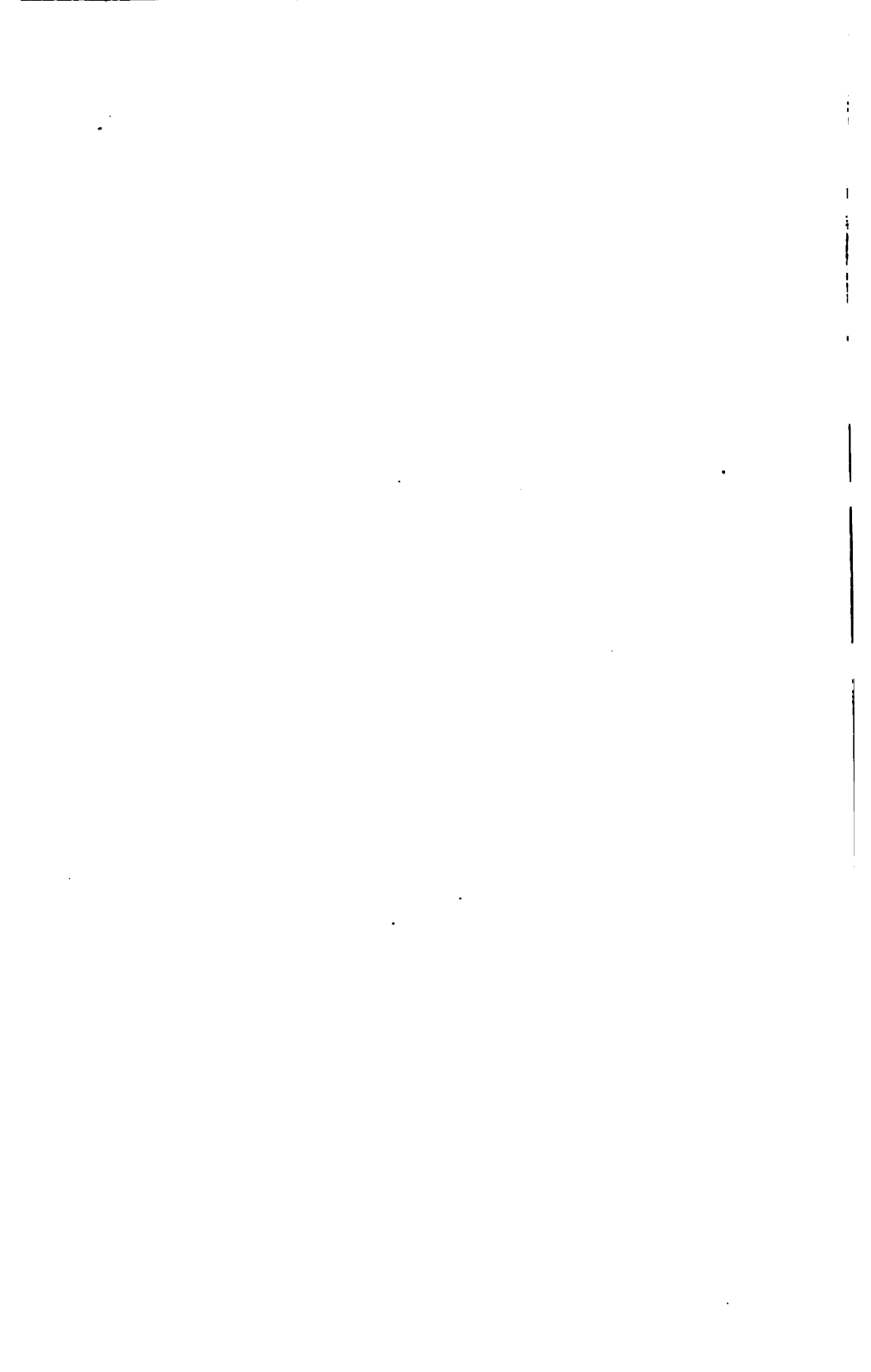
BULLETIN 403. Comparative tests of run-of-mine and briquetted coal on the torpedo boat *Biddle*, by Walter T. Ray and Henry Kreisinger. 1909. 49 pp. 10 cents.

BULLETIN 412. Tests of run-of-mine and briquetted coal in a locomotive boiler, by Walter T. Ray and Henry Kreisinger. 1909. 32 pp.

BULLETIN 416. Recent development of the producer-gas power plant in the United States, by R. H. Fernald. 1909. 82 pp., 2 pls. Reprinted as Bureau of Mines Bulletin 9.

BULLETIN 428. The purchase of coal by the Government under specifications, with analyses of coal delivered for the fiscal year 1908-9, by G. S. Pope. 80 pp. 10 cents. Reprinted as Bureau of Mines Bulletin 11.







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